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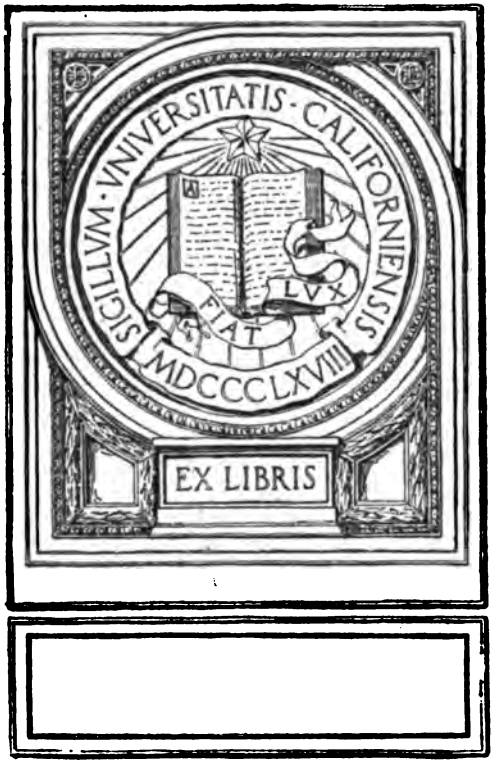

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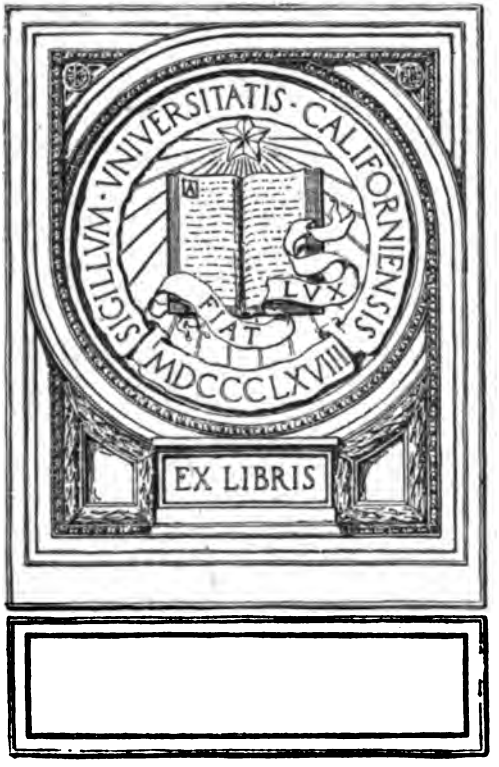

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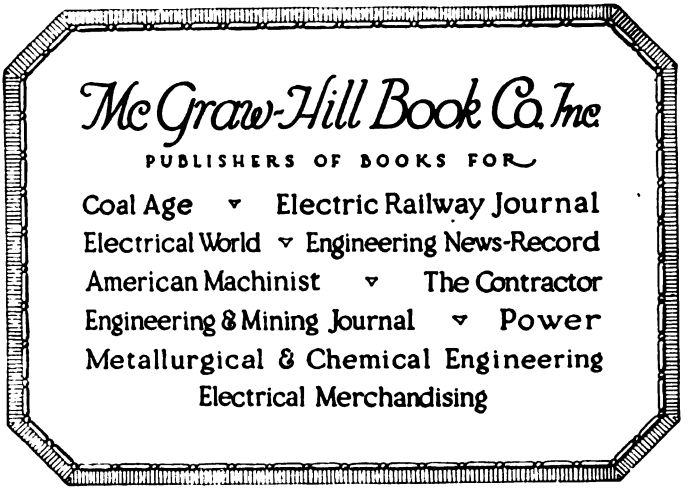
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METALLURGY

OF

LEAD

BY

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PREFACE

Since the last revision of my treatise, "The Metallurgy of Lead and the Desilverization of Base Bullion," physical and chemical research has given precision to the knowledge of the properties of lead, its alloys and compounds, and of many metallurgical reactions; the character of the ores treated has changed; blast roasting has revolutionized the details of the treatment of lead ores; the recovery of values from intermediary products has been modified; the ordinary practice of desilverization of lead bullion has been altered and in part supplemented by electrolysis.

In undertaking to prepare a new edition it was found that by far the larger part of the book had to be rewritten, and the remainder altered to a considerable extent. The present work has become a new book. It retains only the chapters dealing with reverberatory smelting of lead ores and with German cupellation in about their original forms. While the smelting of sulphide lead ores in the reverberatory furnace has become absolete with the advent of blast roasting, the treatment of the intermediary products of a refinery in the furnace has grown in importance. It seemed advisable to retain this chapter which, in addition to the chemistry of the processes, contains much information of practical value. The reason that prompted the retention of German cupellation was that cupellation as a process can not be studied satisfactorily with English cupellation, which has become the general method of working. The parts dealing with receiving, weighing, and sampling of ores have been omitted, as they have been thoroughly discussed elsewhere.

Since 1892 I have reviewed annually in the Mineral Industry the progress that has been made in the metallurgy of lead and lead-silver, and have thus remained in touch with what is going on in this branch of metallurgy. As a study of the technical literature is insufficient for this purpose, the leading lead plants of the United States and Canada have been visited. The importance of such a study becomes evident when the drawings and working data given in the present book are compared with those contained in the preceding edition.

I desire to express my obligation to the American S. & R. Co., Balbach S. & R. Co., Consolidated M. & S. Co. of Canada, Eagle-Picher Lead Co., International S. Co., Northport S. & R. Co., St. Louis S. & R. Co., and

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United States S. R. & M. Co., for the privilege of studying their various plants, and to the officers at the works for the interest they have shown in my endeavors; also to several manufacturing firms for the permission to use a number of their blueprints. In looking at the finished treatise it becomes evident that, without this assistance from the works, the aim to write a book which represents modern practice could not have been attained.

In carrying out calculations, and in seeing the book through the press I have had the assistance of Professors C. R. Hayward, G. A. Roush, and H. L. Seaver.

H. O. HOFMAN.

MASSACHUSETTS INSTITUTE OF TECHNOLOGY,
CAMBRIDGE, MASS., August, 1918

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METALLURGY OF LEAD

CHAPTER I

INTRODUCTION

1. **Historical Notice.**[1]—Lead is one of the six so-called prehistoric metals. The Egyptians[2] used only copper in the first prehistoric period (— –7000 B.C., long chronology); in the second (7000–5000 B.C.), gold, silver, and lead came in together; in the 18th Dynasty (1600–1400 B.C.), lead became very common. One of the oldest pieces of lead extant is probably that of a figure in the British Museum[3] which antedates 3800 B.C.

The Phœnicians, called the greatest metallurgists of antiquity, founded Cadiz about 2300 B.C. They worked the Rio Tinto deposits in which silver-bearing galena occurs in the gossan of the pyrite; their slag-dumps are underlain by older slag-heaps. That they traded for lead is mentioned in Ezekiel, XXVII, 12.

In the library of Ashur-bani-pal (668–626 B.C.) are found many bilingual copies of Babylonian originals from about 2000 B.C.; one of these contains a hymn to the fire-god, "Gibil—of copper and lead their melter art thou, of gold and silver their purifier art thou."[4] At Assur (1300 B.C.) a block of lead with inscriptions, weighing 1,000 lb., was found, beneath which were plates of gold and silver.[5] Lead must have been a well-known metal with the Hebrews, as in Exodus, XV, 10, it is said that the Egyptians sank as lead in the mighty waters.

The Greeks mined lead ores on the islands of Rhodes, Cyprus, and Euboea. The silver-lead mines of Laurium in Attica were in operation before 560 B.C.; they flourished 100 years later, and were considered to be worked out at the beginning of our era. In 1863 they were re-opened by a French

[1] Frantz, A., "Blei und Zinn im Alterthum," *Berg. u. Hüttenm. Z.*, 1880, XXXIX, 365, 437, 449.

Hofmann, K. B., "Das Blei bei den Völkern des Alterthums," Heft 472 of Virchow-Holtzendorff's "Sammlung gemeinverständlicher wissenschaftlicher Vorträge," Habel, Berlin, 1885.

Pulsifer, W. H., "Notes for a History of Lead," Van Nostrand, New York, 1888.

Freise, *Oest. Zt. Berg. u. Hüttenw.*, 1905, LIII, 354, 367, 383, 391, 405, 436.

[2] Petrie, "The Metals in Egypt," "Ancient Egypt," 1915, pt. 1, 12–23; abstr., *Eng. Min. J.*, 1914, XCVIII, 912.

[3] Agricola, G., "De re metallica," 1556; transl. by Hoover, H. C. and L. H., *Mining Magazine*, London, 1912, 390.

[4] Delitsch, F., "Assyrisches Handwörterbuch," Hinrichs, Leipsic, 1896, 176 and 221.

[5] Andrae, *Mittheilungen der Deutchen Orient-Gesellschaft*, June, 1914, No. 54.

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company[1] which erected smelting works and treated new ore and the ancient concentration- and slag-dumps; the works are in operation at present.

The Romans worked lead mines in Sardinia, which had been opened up by the Phœnicians and operated by the Carthaginians, and in southeastern Spain in the third century B.C.; both regions are lead producers today. While the Romans spared the mineral resources of Italy, they carried on mining operations in conquered countries; thus in Gaul they extracted ore at Villefranche, Pontgibaud, Evreux, Lillebonne, and other places; in Britain,[2] in Somersetshire, Derbyshire, Shropshire, etc., where they found slag-dumps proving that the Britons before them had mined and smelted silver-lead ores. In Germany some Roman mines in the region of the Rhine were opened up between 700 and 1000 A.D.; the celebrated mines in the Harz mountains were discovered in 968, those of Saxony, Silesia, and Bohemia about 1200.

At present the United States is the leading lead producer of the world. In this country[3] lead mining dates as far back as 1621, when lead was mined and smelted near Falling Creek, Va. During colonial times lead mines were operated mainly in North Carolina, New York, and the New England states, but only on a small scale, and not very successfully. The mines principally mentioned are the Washington mine, Davidson county, N. C., the Rossie mine, St. Lawrence county, N. Y., and the mines near Middleton, Conn., and Southampton, Mass. None of them were worked continuously, and at present there are produced in the East only about 4,000 tons of lead, of which Oklahoma furnishes over 2,000 tons, the rest coming from Virginia, Tennessee, and North Carolina.

Lead ores are supposed to have been found in Wisconsin in 1682 by Perrot.[4] The lead ores of Missouri were discovered in 1700 by Pennicaut, one of Le Sueur's party, and were first worked successfully in 1720; Mine La Motte, of Madison county, which is worked at present, was the first deposit discovered. In the Wisconsin-Iowa region the first mining was done by Dubuque on the place where the city, named after him, now stands.

From these two districts, the lower and upper Mississippi Valley, came the bulk of the lead of the United States until 1867 when the first great mines of the West were opened. The total reached 205,462 tons in 1915.[5] As the product of the country was 550,055 tons, these districts furnished nearly 38 per cent. of the entire output. The rest came from the western states which

[1] Cordella, *Berg. u. Hüttenm. Z.*, 1864, XXIII, 285, 1865, XXIV, 20.
Lacroix, *Glück auf.*, 1897, XXXIII, 31, *Chem. Z. Rep.*, 1897, XXI, 31.
Georgiades, *Bull. Ind. Min.*, 1897, XI, 523.
Ernst, *Oest. fahrb.*, 1902, L, 50, 447.
Collins, *Eng. Min. J.*, 1904, LXXVIII, 751; 1905, LXXIX, 363; 1909, LXXXVIII, 881.
Guillaume, *Ann. Min.*, 1909, XV, 5; *Eng. Min. J.*, 1909, LXXXVIII, 446.

[2] Hunt, R., "British Mining," Crosby, Lockwood & Co., London, 1884, p. 21.

[3] Ingalls, W. R., "Lead and Zinc in the United States," McGraw-Hill Book Co., Inc., New York, 1908.

[4] Irving, *Tr. A. I. M. E.*, 1880, VIII, 498.

[5] *Min. Res. U. S.*, 1915, I, 187.

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produce argentiferous lead ores, while those from the Mississippi Valley are non-argentiferous, that is, do not contain enough silver to pay for its extraction.

The first discovery of argentiferous lead ore was made in 1863 in Little Cottonwood Canyon, Utah; the first smelting of such ores was carried on in 1865 at Argenta, Mont.; in 1867 similar ores were found at White Pine, Nev., and smelting was begun at Oreana, Nev. In 1869 the mines of Eureka, Nev., discovered in 1864, were opened up, and the treatment of ores was begun in the following year. Next came Utah, where smelteries were erected in 1870, followed by Colorado, which came into prominence in 1878. Later Idaho, New Mexico, and Arizona were added to the list. Colorado for many years was the largest producer of lead ores, but it has been outranked by Idaho and Utah since the exhaustion of the wonderful mines of Leadville after 1890; it furnishes at present only 6–7 per cent. of the country's product, and this high figure is largely due to the central smelting works of Denver and Pueblo which treat more lead ore mined outside of the state than produced in it.

The manner in which prehistoric people extracted lead from ore is unknown. If we recall that oxide and sulphide ores readily give up lead when heated with charcoal, we can imagine that the accidental discovery of this fact may have led to the most primitive furnace, a small pit dug into the ground and enclosed by stones to form a small shaft. If at first such furnaces were worked by natural draft, blast (reeds, bellows) must have been applied to produce the temperature required to form slag. The furnaces (cavity 30 in. deep, side-walls 26 in. high, 2 tuyère-openings) found in Sinai in which copper ores were smelted about 4000 B.C.[1] had this form; the furnaces of Laurium were similar. In early times low shaft furnaces with natural or forced draft formed the apparatus, and charcoal the fuel. Agricola[2] states that in 1556 the inhabitants of the present Austrian province of Carinthia used what may be called a closed stall with inclined bottom for treating galena; they kindled a wood fire and charged the ore upon it, whereupon lead was liberated and, trickling down the hearth, was collected in a basin. Similar contrivances, the Log-furnace and Ash-furnace,[3] were used in smelting ores in the Mississippi valley about 1720. These were in operation until 1836, when the first Scotch ore-hearth was erected. Percy[4] states that in 1730 the ore-hearth was in operation in Derbyshire, England.

The reverberatory furnace[5] fired with coal was first used by Wright in Flintshire, Wales, in 1698. Its use is said to have been suggested by the Spanish priest Alonzo Barba in his book "Arte de los Metales, etc.," published at Madrid in 1640. In the English translation by the Earl of Sandwich of 1669, published by S. Mearne, London, 1674, no such suggestion appears, but even

[1] Flinders-Petrie, W. N., "Researches in Sinai," Dutton & Co., New York, 1906, 242.
[2] *Op. cit.*, 390.
[3] Ingalls, *op. cit.*, p. 96.
[4] "Lead," 216.
[5] Schlüter, C. A., "Gründlicher Unterricht von Hüttenwerken," Meyer, Brunswick, 1738, 110.

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if it had, the text of Barba is so unmetallurgical that his advocacy could amount to little if anything.

The separation of lead and silver is accomplished mainly by cupelling. It does not seem reasonable to suppose that the silver in the earliest times was obtained exclusively from ores containing native silver; the supposition that at least a large part was extracted from argentiferous galena is likelier, and this requires cupellation.[1] The earliest actual reference to this process is probably that found in the hymn to Gibil (p. 1); another is the one in Jeremiah, VI, 29, 30, who lived about 600 B.C.; the remains of Laurium show that cupellation was in operation there. Silver was abundant in Babylonia about 4000 B.C.; this leads to the belief that cupellation had its origin there. The silver vase of Entemena, 2800 B.C., 28 in. high, beautifully wrought, standing on a copper base 7 in. high[2] shows the metallurgical skill of the Babylonians at this early date.

2. Statistics.—The world's production of lead in 1913 is given in Table 1, that of the United States in 1916 in Table 2.

TABLE 1.—WORLD'S PRODUCTION OF LEAD IN 1913[3]
(In metric tons)

Australia	116,000	Japan	3,600
Austria	22,312	Mexico	55,300
Belgium	35,750	Russia	1,000
Canada	17,089	Spain	203,000
France	28,000	Sweden	1,235
Germany	181,000	United Kingdom	48,962
Greece	18,309	United States	396,034
Hungary	1,790		
Italy	21,674	Total	1,142,264

Table 1 shows that the United States produced 34 per cent. of the world's lead. Of the different states given in Table 2, Missouri ranks first in the output of lead ores, and these are non-argentiferous. It is followed by Idaho where is mined silver-bearing galena; next come Utah and Colorado with lead ores containing silver. These four states furnish nearly 90 per cent. of the country's lead. In addition to the primary lead produced from native and foreign ores there has to be considered a considerable amount of secondary or junk lead,[4] that is, lead and lead alloys recovered from scrap, junk, and drosses, which equals from 10 to 15 per cent. of the primary lead produced from ores.

Beside metallic lead there is produced direct from ore a product known as sublimed lead, a mixture of sulphate and oxide of lead.

[1] Benner & Hartmann, "Early History of Cupellation," *Min. Sc. Press*, 1912, CIV, 501.
[2] Handcock, P. S. P., "Mesopotamian Archæology," Putnam, New York, 1912, p. 265.
[3] *Mineral Industry*, 1916, XXV, 456.
[4] Liddell, *Eng. Min. J.*, 1912, XC, 452.
Dunlop, *Min. Res. U. S.*, 1914, I, 17–25.

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TABLE 2.—UNITED STATES' PRODUCTION OF PRIMARY LEAD IN 1916[1]
(In short tons, apportioned according to source of ore)

Domestic ore:		Domestic ore:	
Alaska	659	Washington	217
Arizona	15,328	Wisconsin	3,121
Arkansas	170	Undistributed	96
California	3,633	Zinc residues	5,478
Colorado	33,046		
Georgia	9	Total from domestic ore	596,221
Idaho	170,059	Foreign ore:	
Illinois	670	Africa	328
Kansas	1,737	Canada	1,231
Kentucky	37	Central America	7
Missouri	218,253	Mexico	1,917
Montana	4,961	South America	2,366
Nevada	11,858	Other foreign	236
New Hampshire	46	Foreign lead bullion:	
New Mexico	3,290	Canada	1,072
Oklahoma	10,969	Mexico	11,598
Oregon	9	South America	151
South Carolina	8		
South Dakota	12	Total from foreign ore and lead bullion	18,906
Texas	26		
Utah	111,789		
Virginia	740	Grand total from all sources	615,127

There are sold in the market three grades of primary metallic lead: Desilverized, soft, and antimonial lead. The statistics for 1916[2] are given in Table 3.

TABLE 3.—GRADES OF PRIMARY LEAD OF THE UNITED STATES IN 1916
(In short tons)

Domestic desilverized lead	316,469
Domestic soft lead	165,015
Domestic desilverized soft lead	70,744
Foreign desilverized lead	18,906
Total production of refined primary lead	571,134
Production of antimonial lead	24,038

The average price of desilverized lead in New York in 1916 was 6.858 cts. per pound; that of soft lead which comes from non-argentiferous ore brings from 5 to 10 cts. per 100 lb. less than does the desilverized; the price of antimonial lead varies with the amount of antimony which ranges from 15 to 30 per cent., averaging 16 per cent.

In 1914, with an average price of 3.682 cts. per pound in New York, the total cost of production could not exceed 3.5 cts. per pound if it was to pay expenses.[3]

[1] U. S. Geol. Survey, 1917.
[2] U. S. Geol. Survey, 1917.
[3] Ingalls, *Eng. Min. J.*, 1914, XCVIII, 1073.

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The uses of lead are many. The metal is used in the form of sheet and pipe. Different pieces are joined by burning[1] and soldering.[2] Other forms of metallic lead, used mainly for calking, are lead wool,[3] powdery lead,[4] and ribbon lead.[5] Granulated lead is used for assay purposes. Lead forms a part of important alloys, and serves for making white lead, litharge, red lead, orange mineral, etc.

3. Bibliography.—The leading manuals on non-ferrous metallurgy contain valuable chapters on the metallurgy of lead and lead-silver, as, *e.g.*, those of Stölzel (1863), Kerl (1881), Balling (1885), Phillips-Bauermann (1891), Schnabel-Louis (1905), Hildebrandt (1906), Borchers-Hall-Hayward (1911), Sexton-Primrose (1911), Prost (1912), Austin (1913), Wysor (1913), and Gowland (1918).

The following are the principal publications dealing with lead and lead-silver:

CAHEN, M. "La métallurgie du plomb," *Rev. Univ. Min.*, 1863, XIII, 1, 161; reprint Noblet & Baudry, Paris-Liège, 1863.

GRÜNER, L. "Sur l'état actuel de la métallurgie du plomb," *Ann. Min.*, 1868, XIII, 325; 1869, XV, 519; reprints Dunod, Paris.

PERCY, J. "Metallurgy of Lead," Murray, London, 1870.

RIVOT, L. E. "Traité de Métallurgie," Dunod, Paris, 1873, Vol. II.

EISSLER, M. "Metallurgy of Argentiferous Lead," Crosby, Lockwood & Son, London, 1891.

LAMBERT, TH. "Lead and Its Compounds," Scott, Greenwood & Co., London, 1902.

ILES, M. W. "Lead Smelting," Wiley, New York, 1902.

INGALLS, W. R. "Lead Smelting and Refining with Some Notes on Lead Mining," McGraw-Hill Book Co., Inc., New York, 1906.

HIXON, H. W. "Notes on Lead and Copper Smelting and Copper Converting," McGraw-Hill Book Co., Inc., New York, 1908.

COLLINS, H. F. "Metallurgy of Lead," Griffin, London; Lippincott, Philadelphia, 1910.

Of the works quoted, that of Percy, while old, retains a permanent interest on account of the chapters on the fire-chemistry of lead; the only modern work which aims at completeness is that of Collins.

[1] Rothwell, *Min. Ind.*, 1895, IV, 735.
Lunge, G., "Sulphuric Acid and Alkali," Gurney, London, 1903, I, 433 and foll.
Fay, C. H., "The Art of Lead Burning," D. Williams Co., New York, 1905.
Schoop, *Met. Chem. Eng.*, 1905, III, 200.
Lightfoot, *op. cit.*, 1908, VI, 12.

[2] Richter, C., "Das Löthen des Bleies," Hartleben, Leipsic, Vienna, 1896.

[3] Editor, *Eng. Min. J.*, 1906, LXXXII, 592.
Reinicker, *Gas Age*, 1915, XXXV, 325.

[4] Bailey, U. S. Patent, No. 846384, March 5, 1907, and No. 864443, August 27, 1907.

[5] Editor, *Brass World*, 1910, VI, 173.

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CHAPTER II

PROPERTIES OF LEAD

4. Physical Properties.—Small specimens of native lead have been found in a few localities; all metallic lead is obtained by the reduction of lead compounds.

Lead is the heaviest of all base metals, having according to Reich[1] at 0° C. (water at 4° C. being unity) a specific gravity of 11.37 either cast or rolled; the figure for liquid lead at about 320° C. is 10.645.[2] Commercial lead has a lower specific gravity than 11.37 on account of the impurities it contains.

A freshly cut surface shows considerable luster, but loses it quickly when exposed to atmospheric air.

The color is a bluish-gray.

FIG. 1.—Cast lead, 35d., dendritic forms.

It does not crystallize readily. When cooled slowly, as in the Pattinson process (§219), bundles of small imperfect octahedrons form. Also, when refined lead is poured at the correct temperature into a warm mold and allowed to cool slowly, fernlike isometric crystalline aggregates appear at the surface. Fig. 1[3] shows dendritic forms of lead cast upon a stone; octahedral shapes are seen in Fig. 2[4] which remained in pouring off the liquid portion from slowly cooled molten lead through an opening in the solidified surface. Isometric and monoclinic crystals were obtained by Lehmann[5] in electro-depositing the metal by varying the current density. Other photomicrographs have been published by Cartaud-Osmond[6] and Wäser-Kühnel.[7] It has been proved that there exist allotropic forms of lead.[8] Desch[9] believes

[1] *J. prakt. Chem.*, 1859, LXXVIII. 328.
[2] Vicentini and Omodei, *Ann. Phys. Beibl.*, 1888, XII, 176.
[3] Campbell, Sixth Report Alloys Research Committee, 1904, Appendix.
[4] Campbell, *op. cit.*
[5] *Zt. Kristallogr*, 1889, XVII, 274.
[6] *Rev. Mét.*, 1907, IV, 819.
[7] *Electrochem. Zt.*, 1912, XVIII, 151, 211.
[8] Loutschinky, *Rev. Mét.*, 1909, VI, 986.
Cohen-Inouye, *Zt. phys. Chem.*, 1910, LXXIV, 202.
Creighton, *J. Am. Chem. Soc.*, 1915, XXXVII, 2064.
Heller, *Zt. phys. Chem.*, 1915, LXXXIX, 761.
Cohen-Heldermann, *op. cit.*, 1915, LXXXIX, 733.
Cohen, *Met. Chem. Eng.*, 1915, XIII, 953.
Richards, *J. Am. Chem. Soc.*, 1916, XXXVIII, 2613; *Met. Chem. Eng.*, 1916, XV, 63.
[9] *Eng. Mag.*, 1916, LI, 436.

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that an allotropic change is the cause of the occasional breaking up of lead sheets into separate crystals, especially when in the presence of an electrolyte. Jänecke[1] noticed a transformation at 60° C.

Lead is especially soft when allowed to cool and solidify slowly; it is less soft when cooled quickly, and becomes hard when it contains slight admixtures of other metals, such as Cu, As, Sb, Zn, etc. The grade of commercial lead is often approximately determined by the resistance it offers to scratching with the finger nail, and the facility with which it makes a gray streak on paper.

The tensile strength is low, 2600–3300 lb. per square inch; the elastic limit 0.5 lb. The compressive strength is about 2.5 times the tenacity.

FIG. 2.—Cast lead, reduced, octahedral forms.

It is very malleable, being readily rolled into sheets and hammered into foil. Annealing begins to show its effect at 70° C.,[2] and increases with rise of temperature to 250°–280° C. At 100° C.[3] crystallization occurs to a marked degree in a short time. Recrystallization of cold-worked lead at ordinary temperature takes place very slowly.[4] Figs. 3 and 4 by Campbell[5] represent unannealed strips of lead rolled from small ingots cast in iron and stone molds; they have

[1] *Zt. phys. Chem.*, 1915, XC, 257, 313.

[2] Robin, *Rev. Mét.*, 1913, X, 722, 788.

[3] Ewen-Rosenhain, *Phil. Trans. Roy. Soc.*, 1900, CXCV, A, 279. Beilby, *J. Inst. Met.*, 1911, VI, 2.

[4] Baucke, *Internat. Zt. Metallogr.*, 1912, II, 243.

[5] *Loc. cit.*

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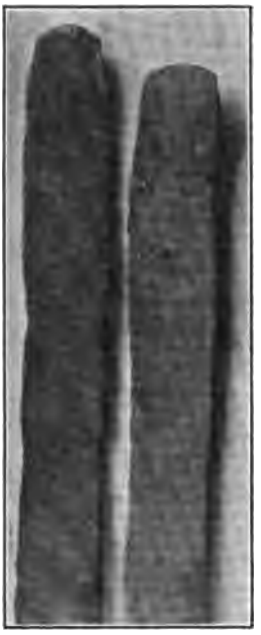
a fine structure. Figs. 5 and 6 are photomicrographs of the same strips annealed for 7 days at about 180° C.; the rearrangement and growth of crystals or grains are well shown. Fig. 7 shows a piece of sheet lead several years old, the small crystals have been formed in the strained metal by the growth of grain at ordinary temperature. The same sheet annealed for 7 days at 180° C., Fig. 8, shows the remarkable growth of crystals taking place at a higher temperature. Lead is not sufficiently ductile to be drawn into fine wire.

The fracture of lead is hackly when broken cold, columnar when hot.

In the form of filings it becomes a solid mass if subjected to a pressure of 13 tons per square inch[1] and liquefies under 33 tons.



FIGS. 3 and 4.—Rolled lead, unannealed.

FIGS. 5 and 6.—Same lead, annealed.

Gases[2] such as O, SO_2, H, N, CO, CO_2, C_xH_y, are insoluble in solid and liquid lead; the gases O and SO_2 may combine with it under suitable conditions, but the others do not.

The melting point of lead is 327.4° C.;[3] at from 3° to 10° below its melting point it becomes very brittle.[4] Its viscosity has been determined by Prüss.[5] The boiling point at 760 mm. is 1525° C.[6] Kahlbaum-Roth-Siedler[7] dis-

[1] Spring, *Ann. Chim. Phys.*, 1881, XXII, 170; *Bull. Soc. Chim.*, 1883, XL, 520.
[2] Stahl, *Chem. Z.*, 1916, XXXIX, 885.
[3] Bureau of Standards, *Bull.* 35, 1915.
[4] Ewen-Rosenhain, *J. Inst. Met.*, 1912, VIII, 2; 1913, X, 119.
[5] *Zt. anorg. Chem.*, 1915, XCIII, 1.
[6] Greenwood, *Tr. Faraday Soc.*, 1911, VII, 145.
[7] *Zt. anorg. Chem.*, 1902, XXIX, 278.

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tilled lead and obtained small globules with adhering cubo-octahedrons; Moissan-Watanabe[1] distilled alloys of Ag-Cu, Ag-Sn, and Ag-Pb, and found that these four metals can be arranged as to volatility in the following order: Pb, Ag, Cu, Sn. Molten lead does not give off any vapor at 500°–520° C.;[2] some vapor may be noticed at 850°–900° C.; the presence of other metals, especially Zn and Sb, reduces the temperature at which vapor forms. The calculations of Richards[3] show that lead is volatile to a small extent at a low red heat (625° C.).

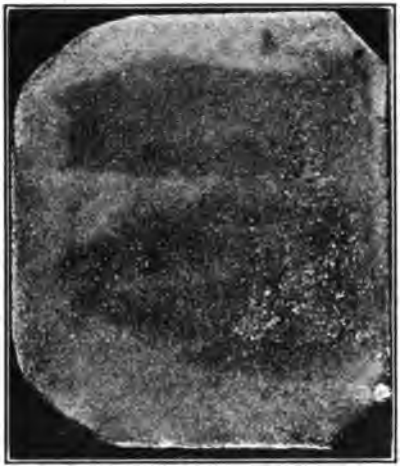

FIG. 7.—Sheet lead, several years old.

FIG. 8.—Same lead, annealed.

The coefficient of linear dilatation of solid lead for 1° C. is 0.2924×10^{-4} at 40° C.[4], and 0.295×10^{-4} at 320° C.;[5] that of the cubical dilatation of liquid lead is 1.29×10^{-4} at 325°–327° C. The mean specific heat of solid lead is $0.02925 + 0.000019t$;[6] the heat in molten lead at its melting point is 11.6 cal.; the latent heat of fusion, 4.0 cal.; the heat in just melted lead, 15.6 cal.; the specific heat in liquid lead 0.042; the latent heat of vaporization 230 Cal. per kilogram; the specific gravity of the vapor 103.5 (referred to H at the same temperature and pressure), or 9.315 g. per cm.^{3} at 0° C. and 760 mm. pressure. The conductivity for heat λ in g.-cal. per cm.^{3} per deg. C., $\lambda_t = \lambda(1+\alpha t)$, is 0.0836 for 0° C., and 0.0764 for 100° C.; or with Ag = 100, that of Pb at 12° C. is 8.5.

The specific resistance in ohms at 0° C. of a wire 1 cm. long and 1 sq. cm. in section is 19.14×10^{-6}; that of a wire 1 ft. long and $\frac{1}{1000}$ in. in diameter is 74.78×10^{-6}; the percentage increase of resistance for 1° C. increase of temperature at 20° is 0.387.[7]

[1] *Compt. rend.*, 1907, CXLIV, 16.

[2] Lewin, *Metall. u. Erz.*, 1913, X, 441.

[3] "Metallurgical Calculations," McGraw-Hill Book Co., Inc., New York, 1908, pt. 3, 558.

[4] Fizeau, *Pogg. Ann.*, 1869, CXXXVIII, 26.

[5] Vicentini-Omodei, *Atti di Torino*, 1887–88, XXIII, 38.

[6] Richards, *loc. cit.*, 557.

[7] "Standard Handbook of Electrical Engineers," 3d Edition, McGraw-Hill Book Co., Inc., New York, 1908, pp. 131 and 134.

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5. Chemical Properties.—The atomic weight is 207.1. Lead undergoes no change in perfectly dry air. If melted in contact with air it is oxidized and becomes covered with an iridescent pellicle said to be Pb_2O; this slowly changes into PbO. If heated to from 300 to 500° C. for sufficient time PbO is converted into Pb_3O_4, which is dissociated into 3PbO and O at 550° C.[1] All oxides with more O than PbO are completely decomposed into PbO and O at 630° C.

Lead is not attacked by water that is free from air;[2] its surface becomes dull by oxidation when in contact with water that is not free from air, or with atmospheric air on account of the water which this contains. The corrosion of lead service-pipes[3] is the greater the softer the water; the presence of O hastens it, that of CO_2 does the same up to a certain point and then retards it, on account of the coating of basic carbonate formed; nitrates, ammonium salts, and calcium and aluminum sulphates assist it.

The best solvent of lead is HNO_3; dilute HCl and H_2SO_4 have little or no action; boiling HCl and H_2SO_4 of 66° Bé. dissolve it slowly. The researches of Lunge[4] and Schmid[5] on the action of H_2SO_4 and HNO_3 on lead of different degrees of purity gave important results. The following are the principal facts: The purer the lead the less will it be attacked by pure or nitrous sulphuric acid up to 200° C., the highest temperature employed under normal conditions in concentrating pans; above 200° C. the action becomes stronger and at 260° C. lead is suddenly dissolved completely, S and SO_2 being formed. This sudden effect of the acid can be slightly retarded by the addition of 1 per cent. Sb and prevented by the addition of 0.2 per cent. Cu to the lead. Concentrated nitrous sulphuric acid acts at all temperatures more powerfully than pure sulphuric acid, and the effect is greater in the presence of air. Dilute nitrous sulphuric acid of a sp. gr. of 1.72 to 1.76 is not as powerful as the pure acid, although if the dilution be continued beyond this point the power increases again instead of

[1] Millbauer, *Chem. Z.*, 1908, XXXIII, 513, 522; 1912, XXXVI, 1436, 1484.
Reinders-Hamburger, *Zt. anorg. Chem.*, 1914, LXXXIX, 71.

[2] Lambeft-Cullis, *Tr. Chem. Soc.*, 1915, CVII, 210.

[3] Wolffhügel, *Arbeiten Kais. Gesundheitsamt*, 1904, II, 112.
Paul-Ohlmüller-Heise-Auerbach, *op. cit.*, 1906, XXIII, 333 (*Zt. Electrochem.*, 1906, XII, 428).
Clowes, *Proc. Chem. Soc.*, 1902, XVIII, 46.
Fordos, *Compt. rend.*, 1873, LXXVII, 1099, 1186; 1874, LXXVII, 1108, 1401.
Clark, "Mass. Board of Health Report," 1898, 541–585; 1900, 487.
Heyn, "Versuche über das Verhalten von Kupfer, Zink, und Blei gegenüber Zement, Beton, und den damit in Berührung stehenden Flüssigkeiten," Ernst and Korn, Berlin, 1911, 59 pages.
Scala, *Mem. accad. Lincei*, 1912(?), VIII, 576 (Abstr. *Am. Chem. Soc.*, 1912, VI, 1271); 1913, XXII, 155 (Abstr. *J. Inst. Met.*, 1913, X, 413).
Vaubl, *Zt. angew. Chem.*, 1912, XXV, 2300.
Gaines, *J. Ind. Eng. Chem.*, 1913, V, 766.
Heap, *J. Soc. Chem. Ind.*, 1913, XXXII, 771, 811, 847.
Lambert-Cullis, *Proc. Chem. Soc.*, 1914, XXX, 198; *J. Chem. Soc.*, 1915, CVII, 210.

[4] *Eng. Min. J.*, 1893, LV, 8, 32, 56.

[5] "Über die Einwickung von reiner, nitroser und rauchender Schwefelsäure und Salpetersäure auf reines Blei und Legirungen von Blei mit Antimon und Kupfer," Riehne, Bâle, 1892; *Zt. anorg. Chem.*, 1892, II, 451, 642.

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diminishing. Junge[1] found that in the matter of corrosion by nitrous sulphuric acid, smoothness or a roughness of surface made more difference than a slight variation in the chemical composition, the rough surface being the more readily attacked.

Sanderson[2] combats the idea that Sb and Cu have a retarding effect upon the dissolution of lead, his experiments in two sulphuric acid plants having shown that even small admixtures of Sb and Cu weakened the lead, and that the corrosion increased with the temperature and the percentage of intermixed alloy.

Lead is attacked by SO_2 between 550 and 850° C. in accordance with the equation $3Pb+SO_2=PbS+2PbO$. It is also acted upon by HF, but the dissolving power is quickly checked by the formation of PbF_2; hence the acid can be stored in lead vessels.[3] Organic acids—acetic, tartaric, and citric acids—attack lead in contact with air.

[1] *Freiberg, Jahrb.*, 1895, II; *Berg. Hüttenm. Z.*, 1896, LV, 33.
[2] *Eng. Min. J.*, 1903, LXXVI, 767.
[3] Jäger, *Zt. anorg. Chem.*, 1901, XXVII, 22.

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CHAPTER III

LEAD OF COMMERCE, ITS IMPURITIES AND THEIR EFFECTS

6. Grades of Lead.—The primary lead produced by smelteries is very pure, containing over 99 per cent. Pb. Of the three grades on the market (§2), the desilverized contains the smallest amount of impurity; next follows the undesilverized; antimonial lead forms a class by itself.

Impure lead bullion (§176), which is to be freed from precious metals either by the Parkes (§231) or the Betts (§290) process, becomes purified to such a degree that the market lead retains usually not more than 0.02 per cent. impurity. Refined lead resulting from desilverization by the Pattinson (§219) process is not sold in the United States although the process is in use in one instance as an auxiliary to the Parkes for the removal of Bi.

Undesilverized lead or soft lead comes from the non-argentiferous ore of the Mississippi Valley. It is less pure than the desilverized, but is of high grade, as it is smelted from ores containing few impurities. The lower the temperature at which the lead is produced, the less will the impurities enter the lead; hence lead from the ore-hearth will be purer than that from the reverberatory furnace, and that from the blast furnace will be the least pure. All the soft lead is liquated and then poled at a temperature a little above the melting point in order to remove impurities. Some smelting plants desilverize the soft lead by means of the Parkes process (§231) in order to improve the character of the metal, as the small amount of silver recovered and the higher price obtained for the lead make the procedure profitable.

Antimonial lead (§262) is a by-product of the Parkes process; it contains from 15 to 30 per cent. Sb. Usually there is present a considerable amount of Cu.

Desilverized and soft leads are used mainly in the manufacture of white lead, of sheet, pipe, shot, and alloys; some minor uses are the preparation of lead wool, powdery lead, ribbon lead, and assay lead. They serve further for the production of litharge, red lead, orange mineral, and lead salts. Antimonial lead is used for making type metal, bearing metal, and other industrial alloys.

The amounts of lead absorbed at present by the different industries is not known. An older statement by Caswell[1] is the following: Paint (white lead, litharge, etc.) 35 per cent.; lead pipe, 20; sheet lead, 8; shot and bullets, 9; solder, 5; tamping, filling knobs, trimmings, etc., 23 per cent.

Table 4 contains chemical analyses of some of the principal American brands of lead. Some well-known European makes have been added for the sake of comparison.

[1] *Iron Age*, 1896, LVII, 9.

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TABLE 4.—CHEMICAL ANALYSES OF COMMERCIAL LEAD[1]

	Joplin, Mo.	Granby, Mo.	St. Joseph, Mo.	Southwest Mo.	Southeast Mo.	Harz Mountains			Pribram	Selby S. & L. Co., Selby, Cal.
	Ore-hearth	Reverberatory furnace	Blast furnace	Ore-hearth	Ore-hearth and Parkes	Reduced in blast-furnace from litharge	Pattinson	Parkes	Luce-Rozan	Parkes
Cu	0.01782	0.05091	0.06400	0.0190	0.0003	0.0600	0.0150	0.00080	0.0024	None
Ag	0.00056	0.00080	0.00007	0.0005	0.0005	0.0028	0.0022	0.00035	0.0018	0.0010
Bi	None	.....	.....	Trace	Trace	Trace	0.0006	.....	0.0023	0.0300
Cd	None	.....	.....	.....	.....	.....	.....	0.00088	.....	Trace
Sn	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.0004
As	0.00011	0.01122	0.00080	Trace	Trace	.....	.....	.....	.....	0.0024
Sb	0.00146	0.00077	0.00130	0.0020	0.0020	0.1340	0.0100	0.00796	0.0056	None
Ni	0.00077	0.00281	0.01600	0.0018	None	0.0050	0.0010	.....	0.0011	0.0001
Co	0.00005	.....				.....	.....	.....	.....	None
Fe	0.00686	0.01582	0.00100	0.0015	0.0015	0.0030	0.0040	0.00100	0.0017	0.0027
Zn	0.00033	0.00090	.....	Trace	0.0004	0.0040	0.0010	0.00092	0.0010	0.0003
Mn	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.0008
S	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Insol	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.0040
Pb, by diff.	99.97204	99.91677	99.91683	99.9752	99.9953	99.7912	99.9662	99.98480	99.9841	99.9579
Reference	*Tr. A. I. M. E.*, 1889–90, XVIII, 687.	*Tr. A. I. M. E.*, 1876–77, V, 324.	Private notes.	H. Garlichs, June, 1915.		*Zt. Berg Hütten Sal. Wes. i. Pr.*, 1870, XVIII, 205.		Private notes.	*Oest. Zt. Berg. Hüttenw.*, 1890, XXXVIII, 497.	*Mineral Industry*, 1907, XVI, 658.

	A. S. & R. Co., Maurer, N. J.		A. S. & R. Co., Omaha, Neb.	Balbach S. & R. Co., Newark, N. J.	International Lead Co., East Chicago		Port Pirie, N. S. W.	Usine Hoboken-lez-Anvers, Belgium	Consol. M. & S. Co., of Canada, Trail, B. C.	U. S. Metals Ref'g. Co., East Chicago, Ind.
	Parkes, Brand B. S.	Parkes, Brand R. L.	Parkes	Parkes	Parkes	Parkes-Pattinson	Parkes	Parkes	Betts	Betts
Cu	None	None	0.0010	0.0010	0.0006	0.0004	None	0.0015	0.0030	0.0010
Ag	0.00053	0.00064	0.0004	0.0003	0.0004	0.0002	0.0009	0.0004	0.001454	0.0007
Bi	0.0366	0.0643	0.0010	0.0790	0.0672	0.0400	None	.....	None	0.0015
Cd	.....	.....	.....	.....	.....	.....	Trace	.....	None	.....
Sn	.....	.....	.....	.....	.....	.....	0.0004	.....	None	Trace
As	None	None	trace	0.0003	0.0010	0.0005	None	0.0005	None	0.0003
Sb	0.0040	0.0084	0.0020	0.0046	0.0062	0.0040	0.0108	0.0030	0.0063	0.0030
Ni	None	None	.....	.....	.....	.....	None	0.0005	None	Trace
Co	.....	.....	.....	.....	.....	.....	Trace	.....	None	Trace
Fe	0.0008	0.0012	0.0008	.....	.....	.....	0.0025	0.0008	0.008	0.0015
Zn	0.0024	0.0036	0.0010	0.0008	.....	.....	0.0001	0.0030	0.0035	Trace
Mn	.....	.....	.....	.....	.....	.....	None	.....	None	.....
S	.....	.....	.....	.....	.....	.....	.....	.....	.....	None
Insol	.....	.....	.....	.....	.....	.....	Trace	.....	.....	Se, Te none
Pb, by diff.	99.95567	99.92186	99.9938	99.9143	99.9246	99.9549	99.9853	99.9903	99.9778	99.9920
Reference	H. H. Alexander, June, 1915.			E. Randolph, June, 1915.	G. P. Hulst, June, 1915.		*Mineral Industry*, 1907, XVI, 658.	*Ann. Min. Belg.*, 1901, VI, 259.	R. H. Stewart, July, 1916.	W. Thum, June, 1915.

[1] Collection of Analyses: Gmelin-Kraut, "Handbuch der Anorganischen Chemie," Winter, Heidelberg, 12, IV, pt. 2, 46.
Mineral Industry, 1904, XIII, 275; 1905, XIV, 388; 1907, XVI, 658.
Treatises quoted on p. 6.

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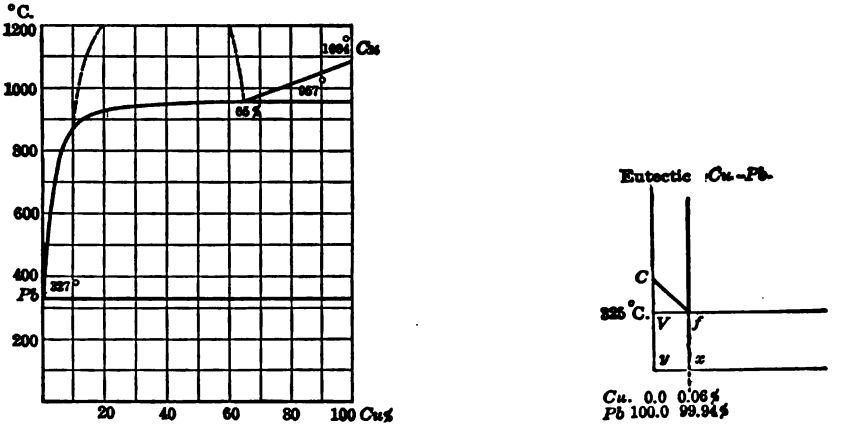
7. Impurities and Their Effects.[1]—In general the analyses given in Table 4 show that the amount of impurity found in the lead of commerce in our days is small, and the number of foreign metals present is large. The harmfulness of these foreign constituents depends on the use to which the metal is to be put. Thus lead which is to be rolled, hammered, or squirted can stand a larger quantity and variety of impurity than that which is to be used in contact with acids; corroding lead which is to be converted into white lead may contain only very small amounts of foreign elements if the paint is to have the required shade of whiteness and the necessary covering power. Details are taken up in §8–22.

8. Copper.—The constitution of the Pb–Cu alloy series has been investigated by Roberts-Austen,[2] Heycock-Neville,[3] Hiorns,[4] Friedrich-Leroux,[5] and Giolitti-Marantonio.[6] In Figs. 9 and 10 of Friedrich-Leroux there are formed in area *fCV* (Fig. 10), upon cooling, crystals of Pb and molten metal; in that of Pb, 65 per cent. Cu (Fig. 9), crystals of Cu, and molten metal.



FIGS. 9 and 10.—Alloy-series Cu-Pb.

Waehlert[7] has shown that if the temperature of 953° C. is raised to 1,025°, the conjugate solutions between 19 and 24 per cent. Cu disappear and form a homogeneous alloy. On the other hand, Bogitch[8] states that the conjugate solutions between 34.5 and 87.0 per cent. Cu disappear at 940° C.

Below the eutectic line, lying at 327° C., and to the left of *fx* (Fig. 10), the alloy consists of crystals of Pb and eutectic; and to the right, of crystals of Cu and eutectic. The eutectic *f* is composed of Pb 99.94 and Cu 0.06 per cent.

[1] Hampe, *Zt. Berg. Hütten. Sal. Wes. i. Pr.*, 1870, XVIII, 195.
[2] "Fourth Report Alloys Research Committee," 1897, 51.
[3] *Philos. Trans. A.*, 1897, XLII, 189.
[4] *J. Soc. Chem. Ind.*, 1906, XXV, 618.
[5] *Metallurgie*, 1907, IV, 299.
[6] Guertler, "Metallographie," Vol. I, pt. 1, 597.
[7] *Metall. u. Erz.*, 1913, X, 578.
[8] *Compt. rend.*, 1915, CLXI, 416.

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Thus it is seen that Cu-bearing lead cooled down slowly will always retain a little Cu. Earlier investigations of Reich[1] had shown that, in liquating, Cu-bearing lead retained as a minimum 0.08 per cent. Cu., while the residual coppery dross formed a brittle porous mass. The steeply rising branch of the curve Pb 65 per cent. Cu (Fig. 9) shows how the solubility of Cu in Pb increases with the temperature, and thereby how important it is to keep low the temperature in melting coppery lead, if the Cu is to be concentrated in the dross collecting on the surface. In order to remove this remaining Cu from Pb an addition of Zn (Parkes process, §231) is necessary which will extract the Cu to the practical limit.

The percentage of copper present in commercial lead does not interfere with the rolling and other mechanical processes. According to Lunge and Schmid,[2] lead with 0.1 per cent. Cu shows the same behavior as pure lead with cold sulphuric acid; so does lead with 0.2 per cent. Cu up to 100° C. At 200° C. lead with from 0.1 to 0.2 per cent. Cu is almost as much attacked as pure lead is, and more than it is by nitrous sulphuric acid. Junge[3] says that in concentrating pans, Parkes lead has been observed to be more readily attacked than Pattinson lead, and he attributes this to the fact that there is less copper in the Parkes lead, which making the melting point lower, increases the liability of being attacked by the acid. If used for corroding or for making flint-glass, the percentage of Cu ought not to exceed 0.0014 per cent.[4] According to Junge the lowest permissible amount is 0.003 per cent.

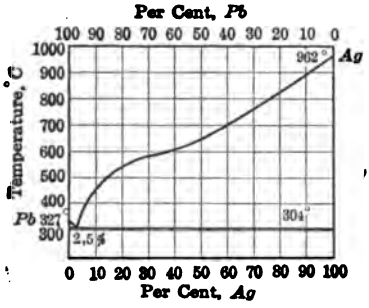

FIG. 11.—Alloy series, Pb–Ag.

9. Silver.—The alloy series Pb–Ag has been studied by Heycock-Neville,[5] Friedrich[6] and Petrenko.[7] The V-shaped curve of Friedrich, given in Fig. 11, shows the existence of an eutectic with 2.5 per cent. Ag, freezing at 300° C., and the extension of the eutectic line to the ordinates; this means that no solid solutions are formed between Pb in Ag, and Ag in Pb. The small amounts of Ag present in market lead, ± 0.1 oz. per ton, are therefore due to eutectic disseminated through the lead; this fact explains in part the irregular distribution of Ag in lead bullion (§176).

What has been said about the effect of Cu in regard to the mechanical treatment of lead holds good for Ag. Small quantities of Ag protect lead against sulphuric acid. Baker[8] says that 1.70 oz. of Ag per ton gives white lead a

[1] *Freiberg. Jahrb.*, 1860, 186.
[2] *Loc. cit.*
[3] *Loc. cit.*
[4] Hampe, *Zt. Berg. Hütten. Sal. Wes. i. Pr.*, 1870, XVIII, 209.
[5] *Phil. Trans.*, 1897, CLXXXIX, 37.
[6] *Metallurgie*, 1906, III, 398.
[7] *Zt. anorg. Chem.*, 1907, LIII, 202.
[8] *Dingl. Polytech. J.*, 1864, CLXXIII, 119.

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reddish tinge, but this is not the case with 0.15 oz. per ton. Landsberg[1] gives as minimum 1.03 oz. for corroding lead; Junge, 0.17 oz.

10. Copper and Silver.—The investigation of Friedrich and Leroux[2] has shown that the alloy series Pb–Cu–Ag forms a ternary eutectic with Pb 97.5, Cu 0.5, Ag 2.0 per cent., which freezes at from 0.5° to 1° C. below the binary eutectic of Pb–Ag.

11. Gold.—While Au is not found in market lead, it occurs in lead bullion. Its behavior in certain processes, for example that of Pattinson, is readily explained by the constitutional diagram traced by Vogel[3] given in Fig. 12. This shows the absence of solid solutions, the presence of two chemical compounds, Au_2Pb and $AuPb_2$, and that of an eutectic, consisting of $AuPb_2$ and Pb, with about 85 per cent. Pb, freezing at 215° C., and undergoing a transformation at

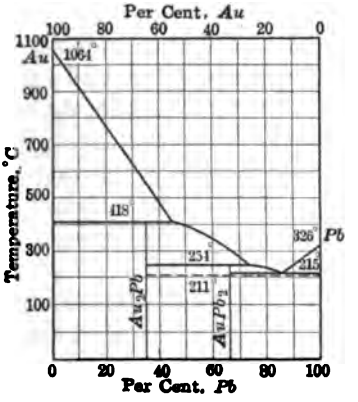

FIG. 12.—Alloy series, Pb–Au.

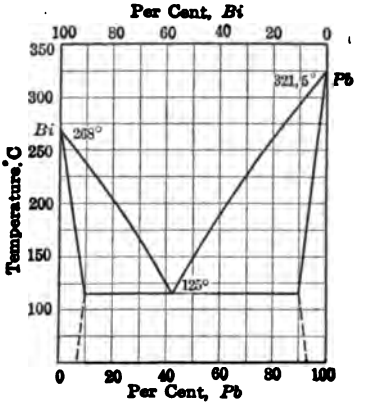

FIG. 13.—Alloy series, Pb–Bi.

211° C. The similarity of this curve with that of the alloy series Pb–Ag at the lead-end indicates that in lead smelting, Au, which forms a series of solid solutions with Ag,[4] will follow the Ag unless other causes interfere with this tendency.

12. Bismuth.—The equilibrium diagram of the Pb-Bi alloy series by Kapp[5] and Stoffel,[6] Fig. 13, shows an eutectic with 44 per cent. Pb, freezing at 124°–125° C., and solid solutions of about 10 per cent. at either end of the eutectic line. Lead is hardened by 0.118–0.352 per cent. Bi,[7] and rendered somewhat crystalline. Junge rolled without difficulty lead containing 2 per cent. Bi into a sheet 10 ft. long, 6 ft. wide, and $\frac{1}{64}$ in. thick; and Burgraf says that lead

[1] *Wagner Jahresber.*, 1875, XXI, 596.
[2] *Metallurgie*, 1907, IV, 293.
[3] *Zt. anorg. Chem.*, 1905, XLV, 17.
[4] Roberts-Austen and Rose, *Chem. News*, 1903, LXXXVII, 2.
Heycock-Neville, *Philos. Trans.*, A, 1897, CLXXXIX, 69.
[5] "Doctorate Dissertation," Königsberg, 1901.
[6] *Zt. anorg. Chem.*, 1907, LIII, 150.
[7] Plattner, *Berg. Hüttenm. Z.*, 1889, XLVIII, 116.

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with 1.920 per cent. Bi can be rolled into sheets as well as refined lead practically free from bismuth.

According to Napier[1] and Bauer,[2] 0.1 per cent. Bi protects lead somewhat from sulphuric acid at 20° C., but not at 100° C., while Junge[3] exposed strips of lead with 0.20 per cent. Bi to sulphuric acid of 60° Bé. at the temperature of the sulphuric acid chamber and found no sensible change. Lunge and Schmid[4] state that lead containing Bi is suddenly dissolved by concentrated sulphuric acid at a temperature below 260° C., and that an addition of 0.02 per cent. Cu counteracts this to some extent. According to Hampe, from 0.0045 to 0.0075 per cent. Bi has no effect on white lead, but Junge says that 0.003 per cent. makes itself felt. Refined lead containing over 0.05 per cent. Bi[5] is no longer classed in this country as corroding lead. Endemann states that bismuth favors the corrosion of lead, a small black residue remaining, containing metallic bismuth.

The reason that the Bi-content of corroding lead is limited in this country to 0.05 per cent., a commercial compromise, is that Bi gives white lead a grayish color, especially where scrap or uncorroded buckles are reset in the stacks. It has also been found that it renders the uncorroded lead more or less spongy and granular, with the result that there is an accumulation of this material in the grooves of the millstones, and their surfaces are likely to become coated. As long, however, as the machinery does its work properly there is no danger of metallic particles contaminating the paint.

There are two ways of removing Bi from Pb: one, the Pattinson process (§219), while imperfect, is sufficient for practical purposes; the other, the Betts process (§290), is perfect, as Bi is not deposited on the cathode, but remains undissolved in the anode mud.

13. **Cadmium.**—This occurs only in traces in market lead. The constitutional diagram by Kapp[6] and Stoffel[7] shows an eutectic with about 80 per cent. Pb, freezing at 249° C., and on the eutectic line a solid solution of lead with about 4 per cent. Cd.

14. **Tin.**—The constitution of Pb–Sn alloys has been the subject of a number of investigations,[8] mainly on account of the formations of solid solu-

[1] *Chem. News*, 1880, XLII, 314; *School Min. Quart.*, 1885–86, VII, 97.
[2] *Ber. deutsch. chem. Gesellsch.*, 1875, CCX, 48; *School Min. Quart.*, 1885–86, VII, 117.
[3] *Op. cit.*
[4] *Op. cit.*
[5] *American Chemist*, 1876, VI, 457; *Wagner Jahresber.*, 1877, XXIII, 422.
[6] *Zt. anorg. Chem.*, 1907, LIII, 152.
[7] Doctorate Dissertation, Königsberg, 1901.
[8] Roberts-Austen, *Engineering*, 1897, LXIII, 223.
Kapp, Doctorate Dissertation, Königsberg, 1901.
Shepherd, *J. phys. Chem.*, 1902, VI, 522.
Stoffel, *Zt. anorg. Chem.*, 1907, LIII, 139.
Rosenhain-Tucker, *Philos. Trans.* A., 1908, CCIX, 89.
Guertler, *Zt. Electrochem.*, 1909, XV, 129.
Degens, *Zt. anorg. Chem.*, 1909, LXIII, 212.
Mazzotto, *Internat. Zt. Metallogr.*, 1911, I, 289; and others.

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tions and the transformations the alloys undergo. A combination of the data of Rosenhain-Tucker, Degens, and Mazzotto is given in Fig. 14. This shows an eutectic with 35.97 per cent. Pb freezing at 181° C.; and a solid solution at either end of the eutectic line, Pb with 18 per cent. Sn, and Sn with 0.37 per cent. Pb. There are further seen two transformations at 161° and 146° C. accompanied by changes in the amounts of Pb and Sn held in solid solution.

Tin makes lead light-gray and hard. It is uncommon in market lead. Lead containing it is more affected by sulphuric acid than pure lead.[1] The effect in corroding has not been studied. It is removed by heating the lead to a bright-red heat with access of air; part of the tin collecting on the surface as oxide is first drawn off as a powder, and the rest as a slag consisting of SnO_2 or $XPbO.SnO_2$ dissolved in PbO. (Softening of Lead Bullion, §241.) Industrial alloys are discussed in §25.

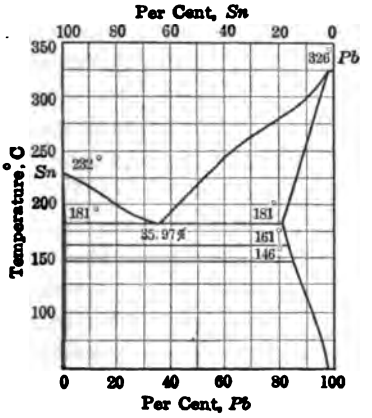

FIG. 14.—Alloy series, Pb–Sn.

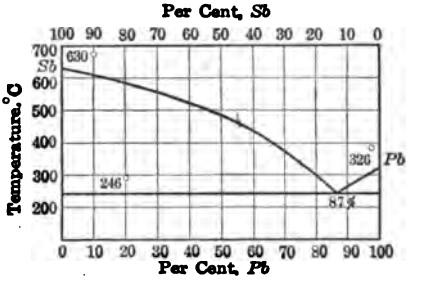

FIG. 15.—Alloy series, Pb–Sb.

15. Antimony.—The constitution of Pb–Sb alloys is simple when compared with those of Sn. It has been studied by Heycock-Neville,[2] Roland-Gosselin.[3] Stead,[4] Charpy,[5] Campbell,[6] Gontermann,[7] and Loebe.[8] Fig. 15 by Gontermann shows an eutectic with 87 per cent. Pb, freezing at 246° C., the eutectic line extending to the ordinates.

Even small quantities of antimony give lead a grayish-white color, and make it harder and less malleable than ordinary lead. A bar of lead containing some antimony will show, especially in the center, an uneven, moss-like surface. Hampe[9] finds that 0.005 per cent. Sb does not harden lead; Heeren[10] states that 0.25 per cent. makes lead hard, but that it is still malleable. Lead with

[1] Napier, Bauer, see §12.
[2] *J. Soc. Chem. Ind.*, 1892, LXI, 904.
[3] *Bull. Soc. d'Encouragement*, 1896, I, 301.
[4] *J. Soc. Chem. Ind.*, 1897, xvi, 200.
[5] "Contributions à l'Etude des Alliages," Chamerat & Renouard, Paris, 1901, pp. 131, 212.
[6] *J. Franklin Inst.*, 1902, CLIV, 205.
[7] *Zt. anorg. Chem.*, 1907, LV, 419.
[8] *Metallurgie*, 1911, VIII, 7.
[9] *Loc. cit.*
[10] Percy-Rammelsberg, "Die Metallurgie des Bleies," Vieweg, Brunswick, 1872, 49.

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0.1 per cent. Sb is not so easily attacked by cold sulphuric acid as pure lead, but more easily by hot acid. Lunge and Schmid[1] substantiate this. They found that an addition of 0.2 per cent. Sb did no harm with cold sulphuric acid, appearing rather to be beneficial; but lead with over 0.2 per cent. Sb was more affected than soft lead, and the discrepancy increased with a higher temperature to an enormous degree.

For corroding, lead may not contain over 0.005 per cent. Sb (Hampe, Landsberg). Junge[2] finds that 0.05 per cent. Sb retards the corrosion, but has no effect on the color of the white lead, while 0.1 per cent. has a decidedly bad influence.

Antimony is removed from lead when this is brought to a bright-red with free access of air; the Sb is oxidized in part to volatile Sb_2O_3 which passes off with the gases, in part to stable Sb_2O_5 which combines with PbO, formed at the same time, to form $Pb_3(SbO_4)_2$, and is held in solution by the molten litharge.

Industrial alloys are discussed in §24.

16. Arsenic.—The lead-end of the equilibrium diagram has been studied by Friedrich[3] and Heike.[4] The curve of Friedrich, Fig. 16, shows an eutectic with 2.5–3.0 per cent. As, freezing at 292° C.; in cooling, layering takes place before any retardation in the cooling curve is noticed. Heike carried on his investigations in a closed porcelain tube heated in an electric furnace; he found the eutectic to contain 3.5 per cent. As and to freeze at 280° C.; he noted the absence of solid solutions and chemical compounds; extrapolation of the curve gave the melting point of As as 852° C.

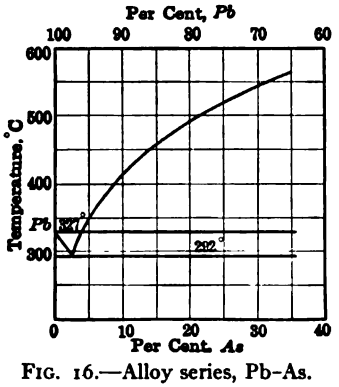

FIG. 16.—Alloy series, Pb-As.

The amount of As found in commercial lead is too small to have any effect upon its mechanical and chemical properties. In refining lead, it is oxidized and slagged as $Pb_3(AsO_4)_2$ before the Sb, with which it is usually associated.

Industrial alloys are discussed in §27.

17. Nickel and Cobalt.—These occur very rarely in market lead. The constitutional diagram of Pb–Ni by Voss[5] shows neither chemical compound nor eutectic, nor any solubility of Ni in Pb. Portevin,[6] however, notes an eutectic with 0.07 per cent. Ni, freezing at 323° C. The alloys of Pb–Cu–Ni have been studied by Parravano-Mazzetti.[7]

The curve of Pb–Co by Lewkonja[8] shows the presence of an eutectic with

[1] *Loc. cit.*
[2] *Loc. cit.*
[3] *Metallurgie*, 1906, III, 46.
[4] *Internat. Zt. Metallogr*, 1914, VI, 49; *J. Inst. Met.*, 1914, XII, 291.
[5] *Zt. anorg. Chem.*, 1908, LVII, 47.
[6] *Rev. Mét.*, 1907, IV, 813.
[7] *Gazz. chim. ital.*, 1914, XLIV², 375; *Chem. Abstr.*, 1915, IX, 1025.
[8] *Zt. anorg. Chem.*, 1908, LIX, 314.

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about 1 per cent. Co, freezing at 326° C., and the extension of the eutectic line to the ordinates. Berthier[1] produced a malleable alloy of Pb with 0.04–0.05 per cent. Ni or Co. Mrázek[2] says that from 1 to 2 per cent. Sb favors the entrance of Ni and Co into Pb; but they rise to the surface when furnace lead is melted slowly, and can be skimmed off easily.

18. Iron.—The research of Isaac-Tammann[3] shows that Pb and Fe have no affinity whatever for one another, and are absolutely immiscible.

Market lead contains not more than 0.07 per cent. Fe,[4] and this amount has no effect upon the mechanical properties. Corroding lead ought not to contain more than 0.003 per cent. Fe.[5]

19. Zinc.—The constitution of these alloys has been studied by Heycock-Neville[6] and Arnemann,[7] and the mutual solubility of the two metals by Rössler-Edelmann[8] and Spring-Romanoff.[9] The equilibrium diagram, Fig. 17, shows that the two metals form one eutectic with 98.5 per cent. Pb, freezing at 317° C. In the neighborhood of 950° C. they form a homogeneous solution.

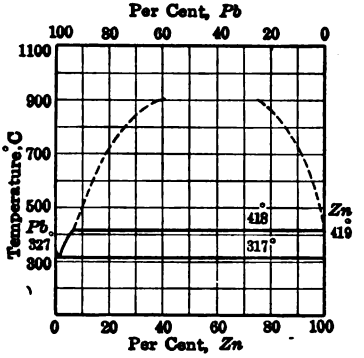

FIG. 17.—Alloy series, Pb–Zn.

The saturation point-curve has been traced only to 900° C., where a separation into saturated and unsaturated solutions is noticed. With fall of temperature the mutual solubility decreases to the melting point of Zn, 419° C., which is lowered 1° by the Zn taking up 0.5 per cent. Pb. At the temperature of 418° there exist conjugate solutions.

Further cooling of the molten metal does not at first lower the temperature, but causes an enrichment in Pb until the liquid contains 96.6 per cent. Pb or 3.4 per cent. Zn; then only does the temperature fall to the eutectic point of 317° C.

[1] "Traité des Essais par la Voie Sèche," Thomme, Paris, 1836, II, 691.
[2] *Berg. Hüttenm. Z.*, 1864, XXIII, 315.
[3] *Zt. anorg. Chem.*, 1907, LV, 59.
[4] Reich, *Berg. Hüttenm. Z.*, 1860, XIX, 28, 224.
[5] Landsberg, *loc. cit.*
[6] *J. Chem. Soc.*, 1897, LXXI, 394.
[7] *Metallurgie*, 1910, VII, 201.
[8] *Berg Hüttenm. Z.*, 1890, XLIX, 245; *Eng. Min. J.*, 1890, L, 573.
[9] *Zt. anorg. Chem.*, 1897, XIII, 29.

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In Table 5 by Rössler-Edelmann are given the amounts of zinc that lead will retain at different temperatures.

TABLE 5.—SOLUBILITY OF ZINC IN LEAD
(Rössler-Edelmann)

Temperature, deg. C.	400	500	600	700
Zinc retained by lead, per cent.....	0.6–0.8	0.9–1.3	1.5–2.3	3.0

Table 6 by Spring and Romanoff shows the mutual solubility of lead and zinc at different temperatures.

TABLE 6.—MUTUAL SOLUBILITY OF LEAD AND ZINC
(Spring-Romanoff)

Temperature, deg. C.	Lower stratum		Upper stratum	
	Per cent. Pb	Per cent. Zn	Per cent. Pb	Per cent. Zn
334	98.8	1.2	.....	.....
419	.....	.....	1.5	98.5
450	92.0	8.0	.....	.....
475	91.0	9.0	2.0	98.0
514	89.0	11.0	3.0	97.0
584	86.0	14.0	5.0	95.0
650	83.0	17.0	7.0	93.0
740	79.0	21.0	10.0	90.0
800	75.0	25.0	14.0	86.0
900	59.0	41.0	25.5	74.5

The data of Table 6 agree in a general way with those of the freezing-point curve, but are very much higher than those of Table 5; the latter have always served as guides to the refiner.

Zinc gives lead a silvery color and makes it so hard that it cannot be rolled; cold and hot sulphuric acid attack it readily. Corroding lead should not contain over 0.003 per cent. (Landsberg). Zinc is removed from lead by heating it to a bright-red, and oxidizing it by admitting air, introducing steam, etc. (See Refining Desilverized Lead, §250.)

20. Manganese.—The freezing-point curve of Williams[1] shows that Pb has no affinity whatever for Mn, and that Mn above its melting point of 1,228° C. does hold some lead in solution. As regards market lead, Mn is present only in very small amounts, and has no practical importance.

21. Aluminum.—The curve of Gwyer[2] shows that Pb and Al have no relations to one another, that Al is insoluble in Pb, and that liquid Al may dissolve some Pb[3]. Antimony is said to favor the union of the two metals.[4]

[1] *Zt. anorg. Chem.*, 1907, LV, 32.

[2] *Zt. anorg. Chem.*, 1908, LVII, 149.

[3] See also Pécheux, *Compt. rend.*, 1904, CXXXVIII, 1042; *Berg. Hüttenm. Z.*, 1904, LXIII, 348; *Compt. rend.*, 1906, CXLIII, 397.

[4] Richards, J. W., "Aluminium," McGraw-Hill Book Co., Inc., New York, 1896, 503.

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In the Rössler-Edelmann process (§231) using aluminized zinc for desilverizing lead bullion, the Al is completely removed in skimming the crusts, as no Al appears in the refined lead.

22. Minor Metals.—The freezing-point curves of lead and the following metals and metalloids are, for the present at least, of little metallurgical importance; Pb–Ca,[1] Pb–Cr,[2] Pb–Hg,[3] Pb–In,[4] Pb–Ir,[5] Pb–K,[6] Pb–Mg,[7] Pb–Na,[8] Pb–Os,[9] Pb–P,[10] Pb–Pd,[11] Pb–Pt,[12] Pb–Rh and Pb–Ru,[13] Pb–Se,[14] Pb–Si,[15] Pb–Te,[16] Pb–Tl.[17]

[1] Harkspill, *Compt. rend.*, 1906, CXLIII, 227.
Donski, *Zt. anorg. Chem.*, 1908, LVII, 210.
Boar, *op. cit.*, 1911, LXX, 1911.

[2] Hindrichs, *op. cit.*, 1908, LIX, 429.

[3] Fay-North, *Am. Chem. J.*, 1901, XXV, 216.
Puschin, *Zt. anorg. Chem.*, 1903, XXXVI, 213.
Jänecke, *Zt. phys. Chem.*, 1907, LX, 399.
Thomson-Keffer, *Met. Chem. Eng.*, 1915, XIII, 367.

[4] Kurnakow-Puschin, *Zt. anorg. Chem.*, 1907, LII, 444.

[5] Deville-Debray, *Ann. Min.*, 1859, XVI, 52.

[6] Smith, *Zt. anorg. Chem.*, 1908, LVI, 137.

[7] Grube, *op. cit.*, 1905, XLIV, 124.
Kurnakow-Stepanow, *op. cit.*, 1905, XLVI, 184.

[8] Mathewson, *op. cit.*, 1906, L, 175.
Roos, *op. cit.*, 1916, XCIV, 329.

[9] Deville-Debray, *Ann. Min.*, 1859, XVI, 9.

[10] Berthier, "Traité des Essais," 1836, II, 681.

[11] Ruer, *Zt. anorg. Chem.*, 1907, LII, 347.

[12] Doerinkel, *op. cit.*, 1907, LIV, 361.

[13] Deville-Debray, *Ann. Min.*, 1859, XVI, 23.

[14] Pélabon, *Compt. rend.*, 1907, CXLIV, 1159.
Friedrich-Leroux, *Metallurgie*, 1908, V, 355.

[15] Tamaru, *Zt. anorg. Chem.*, 1909, LXI, 43.

[16] Fay-Gillson, *Tr. A. I. M. E.*, 1901, XXXI, 527.
Kimura, *Mem. Coll. Sci. Imp. Univ.* Kyoto, 1915, I, 149; *J. Soc. Chem. Ind.*, 1915, XXXIV, 1211; *Chem. Abstr.*, 1916, X, 743.

[17] Kurnakow, *Zt. anorg. Chem.*, 1907, LII, 430; 1909, LXIV, 149; 1913, LXXXIII, 200; 1914, LXXXVIII, 109.
Lewkonja, *op. cit.*, 1907, LII, 454.
Rolla, *Gazz. chim. ital.*, 1915, XLV[1], 185; *J. Inst. Met.*, 1915, XIV, 239.
Roos, *Zt. anorg. Chem.*, 1916, XCIV, 329.

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CHAPTER IV

INDUSTRIAL ALLOYS[1]

23. General.—The alloys in which lead forms the leading constituent are not numerous. They are, however, of considerable importance and at the same time comparatively cheap.

The addition of another metal to lead decreases its softness and malleability. Thus, an addition of Sb or As increases the hardness and impairs the mallea-

TABLE 7.—HARDENING EFFECTS OF SOME METALS UPON LEAD
(Ludwick)

Added metal	Per cent.	Quenched	Annealed	
		Brinell number	Brinell number	For 2 to 3 hr. at deg. C.
Sb	½	7.6– 8.2	6.8– 7.1	
	1	9.8– 9.9	9.5– 9.7	
	2	10.7–10.9	15.1–16.5	230
	4	13.6–13.9	14.0–14.3	
	8	16.8–17.3	15.8–16.1	
Sn	½	6.0– 6.4	6.0– 6.4	
	1	6.8– 6.9	6.6– 7.2	270
	2	8.0– 8.1	7.4– 7.9	
	8	10.6–10.9	11.3–11.4	230
Cd	½	9.1– 9.2	8.9– 9.4	
	1	9.5–10.2	9.7–10.1	270
	2	11.6–12.2	12.6–12.7	
	8	16.7–19.8	14.2–14.5	220
Mg	½	13.5–15.5	13.8–13.9	
	1	17.9–19.6	16.3–16.4	220
	2	22.3–22.6	19.8–20.9	

[1] Charpy, G., "Les Alliages Blancs dits Antifriction" in "Contributions à l'Étude des Alliages," Chalmerot-Renouard, Paris, 1901.

Guillet, L., "Les Alliages Mĕtalliques," Dunod-Pinat, Paris, 1906.

Brannt, W. T., "The Metallic Alloys," Baird, Philadelphia, 1908.

Krupp, A., "Die Legirungen," Hartleben, Vienna, 1909.

Sexton, A. H., "Alloys, Non-ferrous," Scientific Publishing Co., Manchester, 1909.

Kaiser, E. W., "Zusammenstellung der Gebräuchlichen Legirungen," Knapp, Halle, 1911–12; reprint from *Metallurgie*, 1911, VIII, 257, 296.

Hiorns, A. H., "Mixed Metals and Metallic Alloys," Macmillan, New York, 1912.

Ledebur, A.-Bauer, O., "Die Legirungen," Krayn, Berlin, 1913.

Heyn, E.-Bauer, O., "Untersuchungen über Lagermetalle, Antimon-Blei-Zinn Legirungen," *Beiheft Verh. Verein. Beförd. Gewerbefleiss.*, February, 1914.

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bility; one of Sn also increases the hardness, but does not interfere much with the malleability; Cu has a similar effect although not to the same extent, but the tendency of Cu to eliquate permits the use of only a small amount; alloys of Pb–Bi and of Pb–Cd have been prepared, but are little used.

The hardening effects of Bi, Sn, Sb, Cd, Ag, and Mg upon Pb have been studied by Ludwick[1] by means of the Brinell method; some of his results are given in Table 7.

Mg has the greatest hardening effect on account of the chemical compound $PbMg_2$; the hardening powers of Sb, Cd, and Ag are similar; the alloy with 2 per cent. Sb upon annealing has its hardness increased to that of one with ·8 per cent. Sb.

The diffusion of the alloys Pb–Sn, Pb–Bi, Pb–Sb, and Pb–Sn–Sb is shown by Le Gris[2] in an excellent series of illustrations.

Table 8 gives the compositions of some of the leading industrial lead alloys.

TABLE 8.—COMPOSITION OF SOME INDUSTRIAL LEAD ALLOYS

Name	Pb	Sb	Sn	Reference
TYPE METAL				
Type metal	91	9	.....	Brannt, "Metallic Alloys," 368.
Type metal	75	25	.....	Brannt, "Metallic Alloys," 368.
Type metal (*a*)	70	18	10	Brannt, "Metallic Alloys," 368.
Type metal	60	20	20	Brannt, "Metallic Alloys," 368.
Type metal	55	30	15	Brannt, "Metallic Alloys," 368.
Linotype	82	13	5	Wagner, *Brass World*, 1914, X, 83.
Bullets	94	6	.....	Guillet, "Alliages Métalliques," 845.
Engraving plate	60	40	.....	Guillet, "Alliages Métalliques," 845.
Pewter, usual	20	80	.....	Guillet, "Alliages Métalliques," 845.
Pewter, French	18	82	.....	Guillet, "Alliages Métalliques," 845.
BEARING METAL:				
Atlantic Coast Line	85	15	.....	Clamer, *Brass World*, 1906, II, 132.
Pennsylvania R.R.	87	13	.....	Clamer, *Brass World*, 1906, II, 132.
Baltimore & Ohio R.R., thin linings	94–96	3–5	0.5–1.5	Clamer, *Brass World*, 1906, II, 132.
Baltimore & Ohio R.R., thick linings	86	10–12	3–5	Clamer, *Brass World*, 1906, II, 132.
Chicago & Eastern R.R.	84–85	12–14	2	Clamer, *Brass World*, 1906, II, 132.
Chesapeake & Ohio R.R.	91.5	7	1.5	Clamer, *Brass World*, 1906, II, 132.
Magnolia metal	80	15	5	Stanley, "*Metallurgie*," 1906, III, 607.
Solder No. 1	34	.....	66	American markets.
Solder No. 2	50	.....	50	American markets.
Solder No. 3	66	.....	34	American markets.
WHITE METAL OR ANTI-FRICTION BEARINGS:				
Heavy load	87	7	6	Heyn-Bauer, "*Beiheft Verh. Verein. Beförd. Gewerbfl.*," 1914, 225.
Jacoby metal	85	10	5	
French R.R.	80	8	12	
Magnolia metal	78	16	6	
French R.R.	76	10	14	
Ship axle	72	7	21	
French R.R.	70	10	20	
Jacoby metal	62	10	27	
Shot	99.5Pb+0.5As	.....	.....	
	99.8Pb+0.2As	.....	.....	

(*a*) And Cu, 2 per cent.

[1] *Zt. anorg. Chem.*, 1916, XCIV, 161.

[2] *Rev. Mét.*, 1911, VIII, 613.

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24. Lead-antimony.—The freezing-point curve, Fig. 15, shows an eutectic with 87 per cent. Pb freezing at 246° C.; hence, alloys with over 13 per cent. Sb, consisting of Sb and eutectic, must be more brittle than those with less than 13 per cent., as these are made up of Pb and eutectic. Figs. 18–20 represent photomicrographs by Charpy, 2ood., of Pb–Sb alloys with 20, 40, and 70 per cent. Sb; in all of them the light parts represent crystals of Sb imbedded in the dark eutectic.

The industrial Pb–Sb alloys contain less than 24 per cent. Sb. If a higher percentage is to be used, a third metal, usually Sn, has to be added; in many cases Sn replaces part of the Sb.

Tensile, bending, and compression tests of Pb–Sb alloys have been made by Goodman[1] upon three alloys with 10–15 per cent. Sb, and upon one with

FIGS. 18 to 20.—Lead-antimony alloys, × 200.

15 per cent. Sb and 5 per cent. Sn. The tension tests show an elastic limit of about 2,000 lb., and a strength of about 6,000 lb. per square inch; for compression the figures are 2,000 lb. and 14,000 lb. per square inch. Compression tests have been made by Charpy.[2]

The expansion curve of Ewen-Turner, Fig. 21, shows that with additions of Sb to Pb, the expansion increases to a maximum at about 13 per cent. Sb, the eutectic composition, then falls to minimum at about 35 per cent. Sb, rises to a second maximum at about 50 per cent. Sb, falls again to about 60 per cent. Sb, and slowly rises to that of pure Sb. The irregularities cannot be explained by the freezing-point curve.

The hardness curve of the same authors, Fig. 22, given in Shore numbers, shows a uniform rise in hardness to the eutectic point followed by a quicker rise to the maximum with about 75 per cent. Sb, and then a rapid fall to pure Sb.

The tendency to liquation makes the sampling of Pb–Sb alloys always difficult.[3] An industrial method of approximating the Sb-content of an alloy, devised by Richards,[4] consists in casting the alloy to be tested into the form of a

[1] *Engineering*, 1906, XXXII, 376.
[2] *Loc. cit.*, 214.
[3] Nissensen-Siedler, *Berg. Hüttenm. Z.*, 1903, LXII, 421.
[4] *J. Franklin Inst.*, 1899, CXLVII, 398; *Iron Age*, 1899, LXII, May 11, 12.

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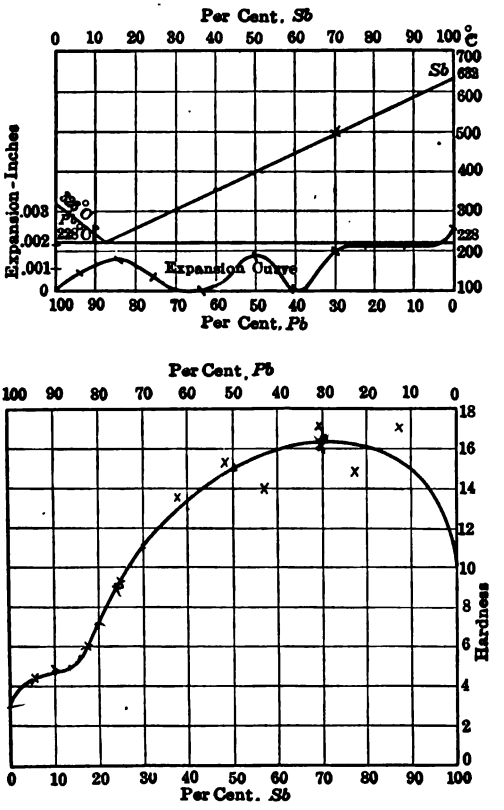
standard size plug in a special mold. This plug is then weighed on a special balance, the graduations on the arm reading directly in terms of Sb–content of the alloy, since the weight of the plug decreases as the Sb–content increases. Friedrich[1] uses another method, by means of which the freezing point of the hard lead to be examined is measured with silica-glass thermometer, and the corresponding compositions read from the melting point curve. The method is technically accurate for alloys with up to 8 per cent. Sb; alloys



FIGS. 21 and 22.—Expansion and hardness curves of Pb–Sb alloys.

containing over 8 per cent. Sb are diluted with a known amount of Pb to fall within the permissible range. An assay takes about 1 min.

Corrosion has been studied in connection with type metal. Meyer-Schuster[2] found that corrosion was less affected by chemical composition than by the manner of casting, of cleaning, and of storing. On the other hand, Zinsberg[3] concluded from his tests that the lower the Pb-content, the greater the resistance to corrosion.

[1] *Metallurgie*, 1912, IX, 446.
[2] *Das Metall*, 1914, I, 352.
[3] *Loc. cit.*

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The leading use of these alloys is for type metal, bearing metal,[1] bullets, and engraving and battery plates.

Type metal usually contains some Sn, as this makes the alloy harder and more rigid without increasing the brittleness; this is of special importance in die-casting.[2] The Pb–Sb alloy is used mainly for quads; however, the following compositions are given by Hiorns[3] as examples of regular type metal: Pb 90, 85, 80, 75 with Sb 10, 15, 20, 25.

A good type metal contains Pb 50–55, Sb 25–30, Sn 25–15 parts. As these compositions are rather high-priced, a mixture of Pb 60, Sb 30, Sn 30 parts is chosen for ordinary type.

Bearing metal demands two constituents, one that is hard to support the load, and one that is soft to act as plastic support for the hard grains and to adapt itself to the form of the revolving surface. Thus, the Pennsylvania R.R. uses the eutectic with its 87 per cent. Pb and 13 per cent. Sb. In many cases Sn is added for the same reasons as in type metal. Details of ternary bearing metals are given in §26.

Bullets contain about 6 per cent. Sb.

Engraving plates are usually composed of Pb 60 and Sb 40 per cent.; often, however, the Pb-content is higher, 80–82 per cent.; the best plates contain some Sn as, *e.g.*, Pb 80, Sb 15, Sn 5 per cent.

Battery plates contain Pb 94, Sb 6 per cent. In order to obtain porous plates for storage batteries, Hannover[4] heats a plate with 96 per cent. Pb to the melting point of the eutectic and separates this by means of a centrifugal machine.

In the preparation of these alloys, the hard lead of the lead refinery forms the base to which, when melted, are added the other components. In starting with the metals, part of the Pb is melted; then Sb is added; this is followed by the rest of the Pb; and lastly is added the Sn. Another method of operating is to melt the Pb, add the Sn, heat the alloy to 500°–600° C., and pour into it the Sb melted in a separate crucible. If Cu is to be added, it is alloyed with the Sb.

25. Lead-Tin.—The freezing-point curve has been given in Fig. 14. The microstructure is similar to that of Pb–Sb alloys except that the Sn-grains are more rounded than those of Sb.

The two metals are easily united in all proportions and form a series of valuable alloys. Some of their mechanical properties have been studied by Sperry.[5] The results of his mechanical tests are given in Fig. 23. A summary of his observations and results is as follows:

Tin and lead combine in all proportions. The color of the alloys ranges from that of pure tin to that of pure lead. In an experiment made on the rolling qualities, it

[1] Clamer, *J. Franklin Inst.*, 1903, CLVI, 49; *Eng. Min. J.*, 1903, LXXVI, 393; *Brass World*, 1914, X, 85.

[2] Lake, *Iron Age*, 1911, LXXXVIII, 532.

[3] *Op. cit.*, p. 348.

[4] *Rev. Mét.*, 1912, IX, 641; *Brass World*, 1913, IX, 323.

[5] *J. Soc. Chem. Ind.*, 1899, XVIII, 113.

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was found that all the alloys can be rolled in the same manner as that employed for rolling tin. The plastic alloys are not so fluid as the non-plastic compositions unless superheated. The yellow color can be produced on the alloys up to and including Sn, 44 per cent.; after this point the lead characteristics begin to predominate. In order to obtain the best results, the metal must be poured at the proper temperature. It was noticed that metal which had been poured "hot," and consequently devoid of the yellow film, became colored on standing exposed to the air for some time. The "tin cry" can be produced from the alloys (starting from pure tin) up to and including Sn 50 per cent.; it is nearly absent, however, in the last few combinations. The strongest alloy in tension is Sn 72.5 per cent., and Pb 27.5 per cent. The strongest alloy in compression is Sn 71 per cent., and Pb 29 per cent. The most ductile alloy is Sn 40 per cent., and Pb 60 per cent. The alloy with the most reduction of

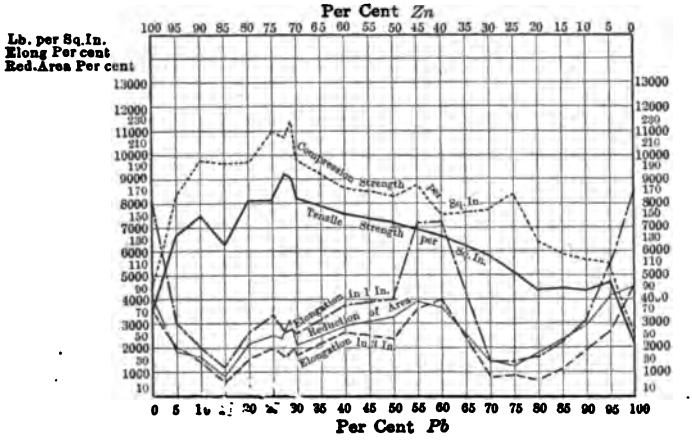

FIG. 23.—Mechanical properties of Pb–Sn alloys.

area is Sn 5 per cent., and Pb 95 per cent. The best alloy for ordinary use is Sn 50 per cent., and Pb 50 per cent., as the surface of the bar is perfectly smooth, and free from the matted surface found on some of the other alloys. The alloys from Sn 15 per cent. to Sn 30 per cent. inclusive, are not homogeneous; the outside of the test-bar fracture showed a fibrous nature, while the core consisted of granular material. The alloys begin to assume a plastic nature at Sn 34 per cent., and end at Sn 15 per cent. The other alloys do not pass through a true plastic state, but pass almost immediately from the solid to the liquid condition. If an attempt is made to use such alloys in place of the plastic compositions, it will be found that the whole mass will be filled with hard lumps which prevent the successful attainment of the end. The alloys between the limits just mentioned are the so-called "wiping solders," and in commerce are known as 3 and 1, and 2 and 1. Munger[1] states that the alloy with 78 per cent. Sn remains plastic for the shortest period of time. The alloys showing the most crystalline nature are those containing from 10 to 20 per cent. of Pb inclusive. The shrinkage more nearly approaches that of tin, and is considerably less than that of lead. The average shrinkage of solder may be said to be 0.06 in. to the foot.

[1] *Met. Ind.*, 1912, X, 290.

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The viscosity of the alloys has been determined by Plüss.[1] The hardness varies with the content of Sn. Thus Sapozhinow[2] finds that the addition of Sn to Pb increases the hardness up to 80 per cent. Sn, and the addition of Pb to Sn increases it up to 30 per cent. Pb; the greatest hardness is reached with 34 per cent. Pb. Stenquist[3] places the greatest hardness between 45 and 50 per cent. Pb.

The solubility of Pb–Sn alloys in acetic acid has been tested by Sackur,[4] who finds, as was to be expected, that the corrosion increases with the Pb-content, both metals going into solution. From the alloy with 10 per cent. Pb there was dissolved in 1 liter of $\frac{1}{20}$ normal acid 7.2 mg. Pb and 0.8 mg. Sn; of $\frac{1}{10}$ normal, 1.9 mg. Pb and 15.5 mg. Sn; of $\frac{1}{4}$ normal, 4.8 mg. Pb and 81.8 mg. Sn. Electrolytic treatment of plates 1 mm. in thickness with NaOH as electrolyte proved successful in that the Sn was dissolved from the lead, leaving behind a spongy residue.

The method of Richards (page 26) of approximating the Sb-content of Pb–Sb alloys is used also for Pb–Sn alloys. The main uses of Pb–Sn alloys are those of pewter, solder, organ pipes, stage-jewelry, toys, etc.

Pewter does not contain over 20 per cent. Pb; the law of France has placed the limit of Pb-content at 18 per cent.

Solder[5] is usually made in three grades,[6] and contains Pb 34 and Sn 66 per cent. (low melting-point); Pb 50 and Sn 50 per cent. (common, solidifies suddenly upon cooling); Pb 66 and Sn 34 per cent. (bent for wiping a joint, as it passes through a pasty stage in solidifying). On account of the ordinary custom of using scrap in the preparation of solder, the presence of from 2 to 5 per cent. Sb is not unusual. Bi is sometimes added to lower the melting-point. The presence of the smallest amount of Zn or Al spoils the solder. An addition of 0.00x per cent. P in the form of phosphor-tin is beneficial; a larger amount is harmful. In Germany (Law of June 25, 1887), solder used for culinary vessels may not contain over 10 per cent. Pb.

Organ-pipes contain 30 per cent. Pb; casket ornaments and stage-jewelry 30–40 per cent. Pb; toys 40–50 per cent. Pb.

In the preparation of the alloys, the Sn is first melted, then the Pb added, and the whole thoroughly stirred.

26. Lead-Antimony-Tin.—It has been shown on page 28 that Pb–Sb bearings are much benefited by the addition of small amounts of Sn. These ternary alloys usually go by the name of White Metal or Antifriction Metal. They are also used in die-casting.[7] The constitutional diagrams of Pb–Sn and Pb–Sb have been shown in Figs. 14 and 15. The freezing-point curve of the

[1] *Zt. anorg. Chem.*, 1913, XCIII, 1.

[2] *J. Russian Phys. Chem. Soc.*, 1908, XL, 92, through *J. Instit. Met.*, 1908, VI, 60.

[3] *Zt. phys. Chem.*, 1910, LXX, 530; *J. Soc. Chem. Ind.*, 1910, XXIX, 281.

[4] *Berg. Hüttenm. Z.*, 1904, LXIII, 319; *Zt. Electrochem.*, 1904, X, 572; *Electrochem. Ind.*, 1904, II, 315.

[5] Lippmann, "History of the Art," *Chem. Z.*, 1911, XXXV, 437.

[6] Editor, *Brass World*, 1907, III, 661; 1911, VII, 193.

[7] Lake, *Mechanical World*, 1914, LVI, 64; *J. Inst. Met.*, 1914, XII, 304.

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Sb–Sn series of alloys, investigated by Reinders,[1] Gallagher,[2] Williams[3] and Loebe[4] is given in Fig. 24. It shows three series of solid solutions accentuated by transformations occurring at 420,° 241° and 231° C. with the lines of division lying at 90, 53 and 8 per cent. Sb.

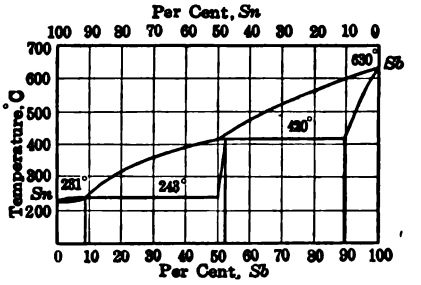

FIG. 24.—Alloy series, Sb–Sn.

The constitution of the ternary-alloy series has been studied by Charpy,[5] Loebe[6] and Heyn-Bauer[7] and certain phases by Campbell-Elder.[8] Figs. 25, 26, and 27 are photomicrographs by Charpy. Fig. 25, representing an alloy with Pb 76, Sn 14, Sb 10 per cent., shows hard white grains of Pb–Sn imbedded in an eutectic; in Fig. 26, standing for the alloy Pb 80, Sn 12, Sb 8 per cent., the hard grains have disappeared; in Fig. 27, representing Pb 70, Sn 10, Sb 20, the white island represents hard grains of Sb.

An adequate discussion of the constitution and properties of this series of alloys is out of place here; only the leading points can be indicated. Since Pb and Sn as well as Pb and Sb form eutectic mixtures, since Sn and Sb form solid solutions, and since Pb forms neither a chemical compound nor a solid solution with either of the other two metals (excepting to a small extent

FIGS. 25 to 27.—Ternary alloys Pb–Sn–Sb. ×200.

with Sn), the crystals separating upon cooling are the same as those of the binary alloys of Sn and of Sb. Loebe holds that there is no ternary eutectic point, while Heyn-Bauer maintain that there are two ternaries. Following Loebe, the eutectic structure is composed of Pb (with perhaps a small amount of

[1] *Zt. anorg. Chem.*, 1900, XXV, 113.
[2] *J. phys. Chem.*, 1906, X, 93.
[3] *Zt. anorg. Chem.*, 1907, LV, 14.
[4] *Metallurgie*, 1911, VIII, 7.
[5] *Loc. cit.*, p. 229.
[6] *Metallurgie*, 1911, VIII, 7.
[7] *Beiheft Verh. Beförd. Gewerbefl.*, Feb. 9, 1914; *Stahl u. Eisen*, 1915, XXXV, 445.
[8] *School Min. Quart.*, 1911, XXII, 244; *Metallurgie*, 1912, IX, 422.

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Sn in solution,) and a solid solution of Sn in Sb or of Sb in Sn. The most fusible alloy is the Pb–Sn eutectic. Owing to the great differences in specific gravity of the Pb crystals and the solid solutions of Sn–Sb, the alloys show a considerable tendency to segregation. The industrial alloys are composed of Pb and solid solutions of Sn–Sb.

The triaxial diagram of Heyn-Bauer, Fig. 28, gives the isotherms of the different alloys, and a line E_2bdH in which the alloy *b* (with Pb 80, Sn 10, Sb 10 per cent.) and the alloy *d* (with Pb 42.5, Sn 53.5, Sb 4 per cent.) represent the ternaries in which the beginning and the end of a fusion, or the solidi-

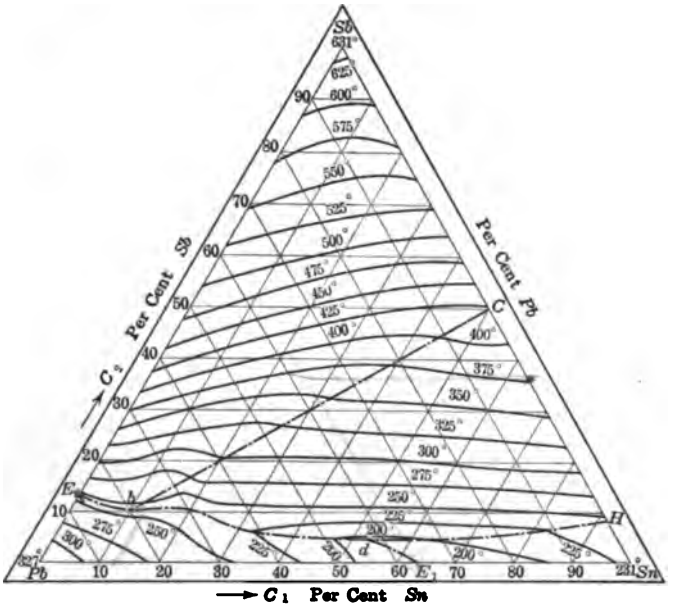

FIG. 28.—Triaxial diagram of Pb–Sn–Sb alloys with isotherms.

fication temperatures, coincide. The line *bd* represents a series of alloys in which melting- and freezing-points approximately coincide. Alloys in which these phenomena are far apart show segregation in slow cooling. This difficulty is in part overcome by additions of from 2 to 3 per cent. Cu, as Cu raises the temperature of first retardation occurring in cooling, while it leaves unchanged the last.

The greatest hardness is reached, according to Heyn-Bauer, with an alloy of the composition Pb 10, Sn 30, Sb 60 per cent. The hardness of Pb–Sb is increased by Sn, that of Sn–Sb by Pb, that of Pb–Sn by Sb. In general, brittleness increases with hardness, except for a series of Sn-rich alloys with less than 25 per cent. Sb, which show toughness greater than that expected from the hardness. Resistance to compression[1] increases with hardness. An addition of Cu increases the hardness as well as the brittleness and especially so with quick cooling.

[1] See also Charpy, *loc. cit.*, p. 230.

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27. Lead-Arsenic.—The freezing-point curve has been given in Fig. 16. The only industrial alloy is shot[1] which contains less than 0.5 per cent. As. Fine sizes contain 0.2 per cent. As, medium 0.3, and coarse 0.35 per cent. The alloys are hard and brittle. Shot is manufactured by pouring the alloy into a perforated iron basin on the top of a tower of considerable height, whence it drops into a vessel filled with cool water which contains a small amount of Na_2S and is covered with grease. The use of centrifugal apparatus and of compressed air for the manufacture of shot are mentioned. The shot from the tower is not uniform, and has to be sorted. This is accomplished by means of a moving round-table over which the spherical globules, fed at the center, will travel quickly, while the misshaped will travel slowly and fall into a separate pocket. In some plants sizing screens, and in others, inclined flat tables have replaced round tables.

28. Other Lead Alloys.—A number of binary and ternary alloys of minor importance than those given above have been put on the market.

Binary Alloys.—Some of these are:

1. *Pb–Bi.*—The alloys are malleable as long as the Pb-content exceeds that of Bi; they are dark gray, and are stronger than Pb. The alloy 1Bi : 2Pb can be rolled into sheets. The alloys on the whole are of little importance.

2. *Pb–Cd.*—A series of alloys with from 2.75 to 18.0 per cent. Cd has been patented by Touceda,[2] which is to serve as bearing metal.

3. *Pb–Mg.*—These alloys[3] form the chemical compound $PbMg_2$, freezing at 521° C., and two eutectics, one with about 68 per cent. Pb, freezing at 460°, the other with about 96 per cent. Pb, freezing at 250° C. An alloy with 1.25 per cent. Mg is said by Moffet[4] to be valuable for antifriction purposes.

4. *Pb–Ni.*—The constitution has been noted in §17. An alloy with Pb 96.5–99.5 per cent. and Ni 3.5–0.5 per cent. has been patented by Allen,[5] which he claims to be stronger and less corrodible than other alloys of similar nature.

Ternary Alloys.—A few of these may be mentioned which have been recently studied: Pb–Ni–Cu;[6] Pb–Sb–Cu;[7] Pb–Sn–Cu;[8] Pb–Sn–Zn;[9] Pb–Sn–Bi;[10] Pb–Ag–Sn;[11] Pb–Cd–Bi and fusible alloys:[12] Pb–Cd–Bi–Sn.[13]

[1] Wettstein, *Met. Ind.*, 1910, VIII, 3.
Scott, *op. cit.*, 1914, XII, 245.
[2] U. S. Patent No. 897431, Sept. 1, 1908.
[3] Grube, *Zt. anorg. Chem.*, 1905, XLIV, 124.
Kurnakow-Stepanow, *op. cit.*, 1905, XLVI, 184.
[4] *Metallurgie*, 1906, III, 226.
[5] U. S. Patent No. 834099, Oct. 23, 1906.
[6] Parravano-Mazzetti, *Gazetta chim. ital.*, 1914, XLIV², 375; *J. Instit. Met.*, 1915, XIV, 234.
[7] Charpy, *loc. cit.*, p. 232.
[8] Carnevali, *Chem. Z.*, 1911, XXXV, 509.
Charpy, *loc. cit.* p. 236.
[9] Levi-Melvano-Ceccarelli, *J. Instit. Met.*, 1912, VII, 278.
[10] Charpy, *loc. cit.*, p. 218.
Sheperd, "Thesis," Ithaca, N. Y., 1903.
Parravano, *Internat. Zt. Metallographie*, 1912, II, 15.
[11] Parravano, *op. cit.*, 1911, I, 89, 1913, III, 15.
[12] Barlow, *Zt. anorg. Chem.*, 1911, LXX, 178.
[13] Stoffel, *op. cit.*, 1907, LIII, 137.
Parravano-Sirovich, *J. Instit. Met.*, 1912, VIII, 322.

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CHAPTER V

LEAD COMPOUNDS

29. Lead Oxide and Carbonate.—*Lead oxide*, PbO (92.83 per cent. Pb, $207Pb+16O=223PbO+50800$ cal., has been found in a vein[1] worked for orpiment; it goes by the name of massicot, and has only mineralogical interest. It is obtained on a large scale as massicot and litharge, which have different physical properties. Massicot, an amorphous yellow powder, is formed by heating lead on a flat hearth to a low red heat, removing the film of suboxide as fast as it forms, and oxidizing it to yellow oxide. If the temperature be raised to the melting-point—that is, to a bright-red heat—and the fused oxide cooled, it solidifies as crystalline litharge. Litharge is obtained on a large scale by cupelling argentiferous lead. It is soft and greasy to the touch and generally crystallizes in orthorhombic octahedrons. Ternier and Le Chatelier[2] have found tetragonal forms. Larson[3] distinguishes red tetragonal massicot and yellow orthorhombic litharge. While molten it is transparent and orange-colored; when cold it is opaque, and its color varies from yellow to red according to the rate at which it has cooled; quick cooling promotes the yellow, slow cooling the red color. Yellow litharge is produced on a large scale by allowing it to run from the furnace over an iron plate and chilling it with water, if necessary; it is thus obtained in small lumps. The red, flaky variety is formed by allowing the running litharge to collect in front of the furnace in cakes of from 1 to 1.5 tons in weight, and to cool slowly. The inner part of a cake will swell up and form flakes of red litharge; the outer and lower parts, having cooled quickly, will remain solid and have a yellow color. This swelling is caused by the giving off of oxygen, which molten litharge absorbs. In solidifying quickly in small lumps the oxygen only makes the surface uneven; in cooling slowly in large lumps the outer solid crust obstructs the passage of the oxygen. This prevents the inner part from solidifying firmly, and causes instead the formation of loose flakes. The flakes and lumps are separated by sifting. Both varieties, when ground, have a reddish-yellow color.

Preparation in this way has no metallurgical interest.

The melting-point is 883° C.[4] or 882°.[5] It is volatile in a current of air

[1] *Mineral. Mag.*, 1914, XVIII, 143.
[2] *Berg. Hüttenm. Z.*, 1895, LIV, 452.
[3] *Am. Mineral.*, 1917, II, 18.
[4] Mostowitsch, *Metallurgie*, 1907, IV, 647.
[5] Schenck-Rassbach, *Ber. deutsch. chem. Gesell.*, 1908, XLI, 2917.

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at 800° C.[1] and very decidedly so at 952°.[2] The dissociation temperature for Pb and O has been calculated as 2348° C.;[3] the specific heat at 23° C. is 0.0519; it is a good conductor of electricity when molten.

Litharge is a strong base and quickly corrodes acid furnace material, with which it forms a silicate. It is an excellent flux, forming fusible compounds with oxides that are infusible alone. They do not always enter into chemical combination with it, but often are simply held in igneous solution by an excess of litharge. Thus fusible mixtures are formed with CaO, BaO, MgO, and Al_2O_3. Table 9[4] gives the amounts of litharge required to form fusible mixtures with the principal metallic oxides.

TABLE 9.—SCORIFYING EFFECTS OF LITHARGE

One part of	Cu_2O	CuO	ZnO	Fe_3O_4	Fe_2O_3	MnO	SnO_2	Sb_2O_3	SbO_2	As_2O_3	As_2O_5
Requires parts of litharge	1.5	1.8	8	4	10	10	12 to 13	Fusible in all proportions.	5	0.4 to 0.8	0.25 to 1

The investigation of Cummingham[5] of the system PbO–CuO up to 70 per cent. CuO has shown the absence of chemical compounds and the presence of an eutectic with 32 per cent. CuO, freezing at 689° C. This explains the well-known fact that, in cupelling, coppery litharge flows more readily than litharge free from Cu.

Litharge readily gives up its O. This is seen from its behavior with S, Te, As, Sb, Sn, Bi, Cu, Zn, Fe.[6]

They become wholly or partly oxidized, and the oxides are either volatilized or scorified by the surplus of litharge, a corresponding amount of lead, which combines with any unoxidized part, having been reduced. PbO begins to act upon Fe at about 925° C.[7]

Reduction by C is noticeable at 400–500°,[8] is very decided at 600°, and more so at 700° C. Reduction by CO to Pb begins at 160–185° C., PbO passing through the state of Pb_2O;[9] Brislee[10] gives 300°. Reduction by H to Pb_2O begins

[1] Doeltz-Graumann, *Metallurgie*, 1906, III, 407.
[2] Hofman-Wanjukow, *Tr. A. I. M. E.*, 1912, XLIII, 543.
[3] Stahl, *Metallurgie*, 1907, IV, 682.
[4] Berthier, "Traité des Essais," I, 513.
Percy, "Lead," p. 16.
[5] *Zt. anorg. Chem.*, 1914, LXXXIX, 418.
[6] Berthier, *loc. cit.*, p. 382.
Percy, *op. cit.*, p. 18.
[7] Friedrich, *Stahl u. Eisen*, 1911, XXXI, 2040.
[8] Doeltz-Graumann, *Metallurgie*, 1907, IV, 420.
Brislee, *J. Chem. Soc.*, 1908, XCIII, 154.
[9] Borchers, *Metallurgie*, 1904, I, 295.
Fay-Seeker-Lane-Fergusion, *Brooklyn Polytech. Engineer*, 1910, X, 7; *Eng. Min. J.*, 1911, XCI, 406.
[10] *Loc. cit.*

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at 211°, and to Pb at 235°;[1] Fay-Seeker-Lane-Fergusion[2] give 190° as the lowest temperature.

Reactions with PbS are taken up in §41.

PbO is slightly soluble in H_2O (1 part in 12,000), readily so in acids and alkali. The best solvent is HNO_3; HF dissolves it sparingly;[3] HCl converts it into $PbCl_2$; H_2SO_4 into $PbSO_4$; H_2S changes it into PbS; HI into PbI_2; alkali solutions convert it into plumbate, *e.g.*, $K_2Pb(OH)_6$. This is stable at 100° C., but is decomposed at a higher temperature: $K_2Pb(OH)_6 + heat = 2KOH + PbO + O + 2H_2O$.

Lead carbonate, $PbCO_3$ (77.5 per cent. Pb; $207Pb + 12C + 48O = 267\ PbCO_3 + 170{,}000$ cal.). This occurs as cerussite. It is a poor conductor of electricity.[4] The white lead of commerce is a basic carbonate. $PbCO_3$ is little soluble in H_2O; the solubility is greatly increased in the presence of CO_2; it is soluble in brine.[5] It is decomposed by heat under the partial pressure of CO_2 at 315° C.;[6] the dissociation temperature in a current of air ought to be lower. The reaction $PbCO_3 + heat \rightarrow PbO + CO_2$ is non-reversible[7] at least in the absence of moisture.

30. Plumbites and Plumbates.—The oxides PbO and PbO_2 form plumbites and plumbates with alkali, alkali eartbs, and some metallic oxides.

Plumbites, $M_2'PbO_2$. Dissolving PbO in an aqueous solution of or in molten KOH forms unstable K_2PbO_2 which is readily oxidized to the plumbate K_2PbO_3 if air is introduced.

Plumbates, $M_2'PbO_3$. There exist two classes: metaplumbates (H_2PbO_3) and orthoplumbates (H_4PbO_4). Of the latter the calcium plumbate, Ca_2PbO_4, has been referred to in connection with blastroasting.[8] It is formed by heating $Ca(OH)_2$ either with PbO_2 or with PbO, air having free access. The compound is stable in dry air free from CO_2. It is decomposed by acid into calcium salt and PbO_2; at ordinary temperature by CO_2 when suspended in H_2O; by boiling with $HNaCO_3$; by H_2O alone at 150° C. The dry salt is decomposed between 950 and 1020° C.[9]

31. Lead Ferrite.—It is known that in the crucible assay the presence of Fe_2O_3 acts unfavorably upon the yield of Pb. The usual interpretation is that Fe_2O_3 oxidizes some of the reducing agent; Kohlmeyer[10] on the other hand attributes it to the formation of a compound of Fe_2O_3 and PbO. Investigations of the system PbO–Fe_2O_3 leads him to the conclusion[11] that there exist

[1] Glaser, *Zt. anorg. Chem.*, 1903, XXXVI, 13.
[2] *Loc. cit.*
[3] Jäger, *Zt. anorg. Chem.*, 1901, XXVII, 22.
[4] Kiliani, *Berg. Hüttenm. Z.*, 1883, XLII, 235, 375.
[5] *Met. Chem. Eng.*, 1916, XIV, 31.
[6] Friedrich, *Stahl u. Eisen*, 1911, XXXI, 1909.
[7] Debray, *Compt. rend.*, 1878, LXXXVI, 513.
[8] Borchers, *Metallurgie*, 1905, II, 1.
[9] Le Chatelier, *Compt. rend.*, 1894, CXVII, 109.
[10] *Metallurgie*, 1910, VII, 289.
[11] *Op. cit.*, 1913, X, 447, 483.

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probably five chemical compounds, $3PbO.Fe_2O_3$; $3PbO.2Fe_2O_3$; $3PbO.3Fe_2O_3$; $2PbO.3Fe_2O_3$; and $PbO.2Fe_2O_3$, all of which are decomposed upon fusion and form ternary mixtures of PbO, FeO, and Fe_2O_3. Of these at least one chemical compound $PbO.FeO.4Fe_2O_3$ is believed to be formed. The melting-point of PbO is depressed 133° C. through the addition of 12 per cent. by weight of Fe_2O_3.

32. Lead Silicates.—There occur two silicates, barysilite, $3PbO.2SiO_2$, and alamosite, $PbO.SiO_2$, which have only mineralogical interest. The constitution of lead silicates was first studied by Simmons[1] and Manihot-Kieser,[2] and later by Mostowitsch.[3] The last concluded that there existed no real compounds, and that the silicates were solutions of PbO in lead glass. This position has been controverted by the researches of Cooper-Shaw-Loomis,[4] Hilpert-Weiler[5] and Hilpert-Nacken.[6] The curve of Hilpert-Nacken, Fig. 29, shows three chemical compounds, $2PbO.SiO_2$, $3PbO.2SiO_2$, and $PbO.SiO_2$, which have been further established by optical analysis,[7] and three eutectics; one of PbO–$2PbO.SiO_2$, freezing at 717° C.; a second, $2PbO.SiO_2$–$3PbO.2SiO_2$; and a third, $3PbO.2SiO_2$–$PbO.SiO_2$, both of which freeze at 670° C. The union of the components[8] begins at 710°, is slow at 750°, and increases rapidly with the temperature. Mostowitsch[9] has shown that mixtures of PbO and SiO_2 ranging in composition from $6PbO.SiO_2$ to $PbO.SiO_2$ begin to soften at temperatures lying between 700 and 750° C., that is, considerably below the melting-point of PbO, which lies at 883°; also that the melting points of lead glasses in general are lowered as the SiO_2-content is increased. Thus the subsilicate $4PbO.SiO_2$ is completely liquefied at 726°; the singulosilicate, $2PbO.SiO_2$, forms a viscous liquid at 724°, and requires 940° C. to flow readily. These data do not quite agree with those to be read from the curve.

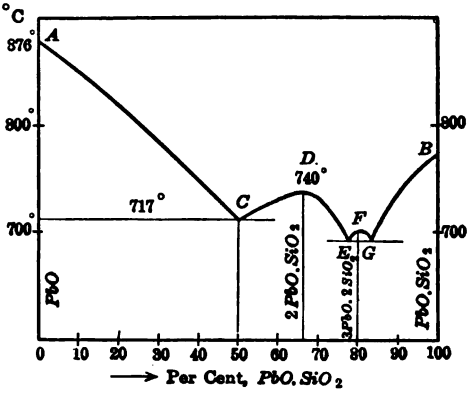

FIG. 29.—Freezing point curve of system PbO–SiO_2.

The fusible silicates are yellow and become darker in proportion to the quantity of PbO they contain. They change their color if they are contami-

[1] *J. Chem. Soc.*, 1903, LXXXVIII, 1449.
[2] *Liebig Ann. Chem.*, 1905, CCCXLII, 356.
[3] *Metallurgie*, 1907, IV, 648.
[4] *Am. Chem. J.*, 1909, XLII, 461.
[5] *Ber. deutsch. Chem. Gesell.*, 1909, XLII, 2969.
[6] *Op. cit.*, 1910, XLIII, 2565; *Metallurgie*, 1911, VIII, 157.
[7] Cooper-Kraus-Klein, *Am. Chem. J.*, 1912, XLVII, 273.
[8] Hilpert, *Metallurgie*, 1908, V, 535.
[9] *Metallurgie*, 1908, V, 535; *Tr. A. I. M. E.*, 1916, LV, 744.

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nated with other metallic oxides, as can be seen if lead is slagged in a scorifier; *e.g.*, Fe colors brown; Cu, green; Mn, purple-black; Ni, brownish-yellow; Co, blue; Te, yellowish-red, the colors growing dark in proportion to the oxide added.

Hilpert[1] heated mixtures of the components severally to 930° and 1,000° C., and found that at 930° only $2PbO.SiO_2$ showed loss by volatilization; that at 1,000° again $2PbO.SiO_2$ gave the greatest loss and was followed by $PbO.SiO_2$; and that mixtures with higher percentages of SiO_2 experienced no loss whatever.

Lead silicates are much used for glazing tiles, pottery, etc. Their behavior has been studied by Seger and Cramer.[2]

The lead from silicates is not readily liberated by the ordinary reducing agents. Sulphur decomposes the singulo-silicate to some extent, but it has less effect on the bi-silicate; FeS throws down some Pb, a double silicate of lead and iron being the result; C reduces from a bi-silicate part of the Pb. In order to extract all the Pb it must be first set free from its combination with SiO_2 by a basic flux; thus Fe decomposes all fusible lead silicates at a bright red heat, provided enough is added to form a singulo-silicate.

The singulo-silicate and bi-silicate of lead are readily decomposed by nitric acid, the tri-silicate is not completely decomposed; the more acid the silicate the less is it soluble.

33. **Lead Borates.**—Fusions of PbO and B_2O_3 give glasses which are colorless or light yellow, and increase in hardness with the content of B_2O_3. Le Chatelier[3] prepared the compound $PbO.3B_2O_3$ by fusion of the components and removal of the excess of B_2O_3 with H_2O. According to Guertler[4] the oxides PbO and B_2O_3 are only partly miscible at temperatures below 620° C.; fusions with 0—0.6857 g. PbO : 1g. B_2O_3 form emulsions; with over 0.06857g. PbO they form glasses.

34. **Lead Chloride, $PbCl_2$** (74.5 per cent. Pb; $207Pb+71Cl=278$ $PbCl_2+83{,}900$ cal.).—It is formed in the dry way by the action of Cl upon Pb, and in a chloridizing roast of Pb and its compounds. It is crystalline or amorphous depending upon the mode of preparation; melts at 498° C.;[5] boils at 956°;[6] has a specific heat of 0.0778 between 260 and 498° C., and of 0.121 between 498° and 580°; a specific resistance ranging from 1395 to 1941 ohms between 498° and 608°. It is soluble in boiling water[7] from which it crystallizes in rhombic crystals upon cooling; is slightly soluble in cold water, less so in water containing a little HCl, the solubility increasing with the content of HCl; soluble in alkaline

[1] *Loc.-cit.*

[2] *Thonindustrie Z.*, 1893, XVII, 1224; *Berg. Hüttenm. Z.*, 1894, LIII, 11.

[3] *Bull. Soc. Chim.*, 1899, XXI, 35.

[4] *Zt. anorg. Chem.*, 1904, XL, 225.

[5] Goodwin-Kalmus, *Phys. Rev.*, 1909, XXVIII, 1.

[6] Weber, *Zt. anorg. Chem.*, 1899, XXI, 305.

[7] Auerbach-Pleissner, "Ueber die Löslichkeit einiger Bleiverbindungen in Wasser," Springer, Berlin, 1907.

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chlorides. It is reduced by H below its melting-point, by heating with $Ca(OH)_2$ and charcoal, or with black flux (K_2CO_3+C), or H_2O-vapor and C.

It forms many double salts. The system $PbCl_2$–KCl shows three chemical compounds,[1] $PbCl_2.KCl$, $PbCl_2.2KCl$, $PbCl_2.4KCl$; and 2 eutectics between KCl and $2PbCl_2.KCl$, and $2PbCl_2.KCl$ and $PbCl_2.2KCl$. The system $PbCl_2$–$Mg(Ca)Cl_2$ has been studied by Menge.[2] $PbCl_2$–$MgCl_2$ forms an eutectic with 91 per cent. $PbCl_2$, freezing at 455° C., the melting point of $PbCl_2$ being 496° and that of $MgCl_2$ 710°. $PbCl_2$–$CaCl_2$ forms an eutectic with 83 per cent. $PbCl_2$, freezing at 467°, the melting point of $CaCl_2$ being 777°. In both cases the eutectic line extends to the ordinates.

Of the oxychlorides Ruer[3] has established the compounds $PbCl_2.PbO$, $PbCl_2.2PbO$, and $PbCl_2.4PbO$. The compound $PbCl_2.PbO$ is decomposed at 524° C., and fuses at 615°; $PbCl_2.2PbO$ fuses at 693° without decomposition; the same is the case with $PbCl_2.4PbO$ at 711°. The basic $PbCl_2.2PbO$ forms in part solid solutions with the other two chlorides.

According to Berthier,[4] $PbCl_2$ and PbS form fusible mixtures which show a strong tendency to percolate through a clay crucible. Truthe[5] has shown that there exists an eutectic in the system $PbCl_2$–PbS with 22 per cent. PbS which freezes at 441° C.

The volatility of $PbCl_2$ has been made the basis of a process for the extraction of Pb from silicious oxide ores.[6]

35. Lead Bromide and Iodide, $PbBr_2$ and PbI_2 (56.41 and 44.92 per cent. Pb).—The compounds are of little importance at present in the treatment of lead ores. According to Mönkemeyer[7] $PbCl_2$ melts at 495° C., $PbBr_2$ at 370°, and PbI_2 at 358°. The compounds $PbCl_2$ and $PbBr_2$ form solid solutions; $PbCl_2$ and PbI_2 are slightly soluble in one another and form an eutectic with 23 per cent. $PbCl_2$, freezing at 306°; $PbBr_2$ and PbI_2 show a similar behavior, being only slightly soluble and forming an eutectic with 51 per cent. $PbBr_2$, freezing at 256° C. The melting-point given for $PbBr_2$ appears to be low; Goodwin-Kalmus[8] give 488° C. The boiling point of $PbBr_2$ is 921°;[9] that of PbI_2 lies between 861 and 954°.[10]

36. Lead Fluoride, PbF_2 (84.48 per cent. Pb; $207Pb+38\ F=245\ PbF_2+101{,}600$ cal.).—It is formed by the union of Pb and F_2, or the action of HF upon $Pb(OH_2)$ or $PbCO_3$. It is a white powder practically insoluble in H_2O and HF, slightly soluble in HNO_3 and HCl, readily fusible to a yellow liquid,

[1] Lorenz-Ruckstuhl, *Zt. anorg. Chem.*, 1906, LI, 71.
[2] *Zt. anorg. Chem.*, 1909, LXIV, 227.
[3] *Op. cit.*, 1906, XLIX, 365.
[4] "Traité des Essais," II, 685.
[5] *Zt. anorg. Chem.*, 1912, LXXVI, 161.
[6] Wells, *Min. Sc. Press*, 1917, CXIV, 507.
Ralston-Williams-Udy-Holt, *Bull. A. I. M. E.*, August, 1917, p. 1205.
[7] *Neues Jahrb. Mineralogie, Beilageband.*, 1908, XXII, 1.
[8] *Phys. Rev.*, 1909, XXVIII, 1.
[9] Helfenstein, *Zt. anorg. Chem.*, 1900, XXIII, 255.
[10] Carnelly, *J. Chem. Soc.*, 1880, XXXVII, 126.

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unchanged at a higher temperature, decomposed at an elevated temperature by H and S, in the cold by H_2SO_4.

37. Lead Fluosilicate, $PbSiF_6$ (59.31 per cent. Pb).[1]—It is formed by the action of H_2SiF_6 upon PbO when some SiO_2 is precipitated; crystallizes from solution with 4 mol. H_2O; dissolves at 15° C. in 28 per cent. of its weight of H_2O, forming a syrupy solution of 2.38 sp. gr.; melts at 60° C. with water of crystallization. Heating a neutral solution causes a partial decomposition into insoluble basic salt and free H_2SiF_6.

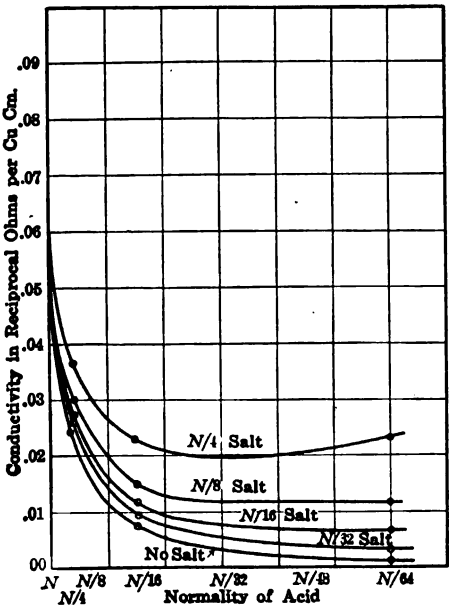

FIG. 30.—Electric conductivity of H_2SiF_6 of different normalities of $PbSiF_6$ dissolved in acid.

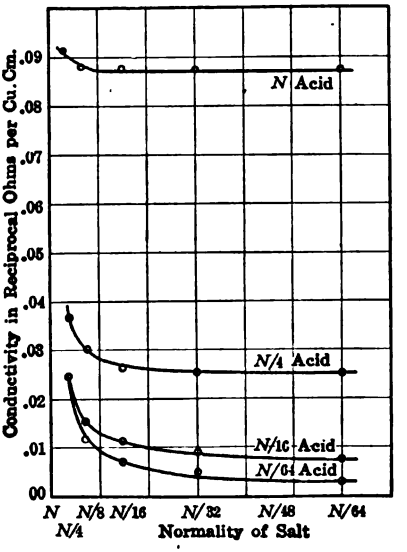

FIG. 31.—Electric conductivity of $PbSiF_6$ when dissolved in different normalities of H_2SiF_6.

The electric conductivity of H_2SiF_6 and of $PbSiF_6$ has been studied by Pattinson.[2] His results are summarized in Figs. 30 and 31; they show at a glance the great effect free acid has in increasing the conductivity of the electrolyte.

Electrolysis does not decompose H_2SiF_6 simply into H and $HSiF_6$; there exists a tendency toward the decomposition into SiO_2 and HF which combine again under the influence of circulation and diffusion.

38. Lead Selenide and Telluride.—*Lead selenide*, PbSe (72.3 per cent. Pb; 207Pb+79 Se=286 PbSe+17,000 cal.) occurs as clausthalite; its freezing-point curve has been noted in §22. It is formed by direct union of com-

[1] Betts, *Tr. A. I. M. E.*, 1904, XXXIV, 175.

[2] *Met. Chem. Eng.*, 1913, XI, 670.

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ponents[1] or by reduction of selenate with H, C, or Al; is readily sublimed; is decomposed by roasting.

Lead telluride, PbTe (62.3 per cent. Pb; 207Pb+126 Te=333 PbTe+ 333 cal.) occurs as altaite; its freezing-point curve has been noted in §22. It is formed by direct union of the elements, is readily sublimed, and decomposed by roasting.

Selenides and tellurides are found in the anode mud formed in the electrolytic refining of copper and lead. They deserve consideration, as the mud forms the raw material for the production of Se, and the Te, for which there is no market at present, is the bane of the refiner in producing fine silver or doré silver.

39. Lead Sulphide, PbS (86.6 per cent. Pb, 207Pb+32S=239PbS+ 20,200 cal.).—This occurs as galena. The existence of subsulphides (Pb_2S, Pb_4S) believed in by older metallurgists[2] and doubted by Percy[3] has been disposed of by Friedrich-Leroux,[4] whose freezing-point curve is given in Fig. 32. This shows two unbroken lines; the crystallization of PbS begins at 1103° C. and solidification of the mixture is completed at 326°, the melting-point of Pb.

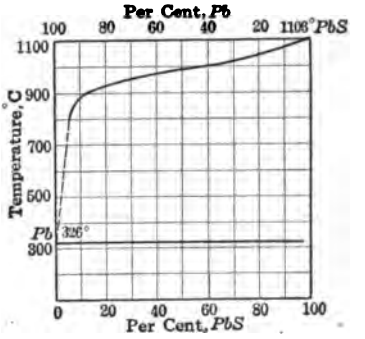

FIG. 32.—Freezing point curve PbS–Pb.

Rössler[5] found that PbS dissolved in Pb would crystallize, upon cooling, in cubes arranged in step-form, resembling those which are frequently seen in wall- and hearth-accretions of blast-furnaces.[6]

PbS is formed in the dry way by heating Pb and S, or PbO with an excess of S, or by reducing $PbSO_4$ with carbonaceous matter.[7] The reduction of $PbSO_4$ is taken up in §40. In the wet way PbS is prepared by precipitation from solution with H_2S. The sulphide produced in the dry way has the same properties as galena; precipitated amorphous PbS is rendered crystalline by heating with exclusion of air.

The melting-point is given as ranging from 935°[8] to 1120° C.;[9] the last figure is the one generally accepted. The sulphide is very fluid when melted,

[1] Rössler, *Zt. anorg. Chem.*, 1895, IX, 41.

[2] Bredberg, *Erdmann J. ökon. Chem.*, 1829, V, 237, 1831; XII, 287; *Poggendorf Ann.*, 1829, VII, 268.

[3] "Lead," 39.

[4] *Metallurgie*, 1905, II, 536.

[5] *Zt. anorg. Chem.*, 1895, IX, 41.

[6] Illustrated in Plattner, C. F., "Die Metallurgischen Röstprocesse," Engelhardt, Freiberg, 1856, p. 201.

[7] Boudouard, *Bull. Soc. Chem.*, 1901, XXV, 284.

[8] Lodin, *Compt. rend.*, 1895, XX, 1164.

[9] Friedrich, *Metallurgie*, 1908, V, 23.
Biltz, *Zt. anorg. Chem.*, 1908, LIX, 273.

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and penetrates the firebrick of the furnaces in which it is treated; often a network of small veins of bright crystalline galena is found in furnace-linings. It is volatile below its melting-point. Doeltz-Graumann[1] heating galena in an atmosphere of N found that sublimation took place at 860° C., while Biltz[2] noticed it only at 950°. A more recent research of Schmehl[3] gives 600° as the temperature at which PbS is partly volatilized. On the walls of blast-furnaces crystals of sublimed galena are of common occurrence. The specific heat of crystallized galena between 0° and 300° C. is 0.0520.[4] Galena is a good conductor of electricity;[5] numerical data are given by Guinchant[6] and Aubel.[7]

The relations existing between PbS and other metallic sulphides are discussed under Constitution of Matte, §180.

PbS is decomposed in the dry way by heating to 600° C. in a current of H,[8] the reaction $PbS+H_2 \rightleftarrows Pb+H_2S$ being reversible; by roasting; and by fusion with a metal which has a stronger affinity for S than has Pb.

In roasting, oxidation begins at 360° to 380° C. according to the writer's experiments with 100-mesh galena. The figures of Friedrich,[9] 554° and 847° C. as the ignition temperaturers in O for particles of 0.1 mm. and over 0.2 mm. in size, appear to be too high. Plattner[10] states that in roasting finely ground galena carefully at a temperature sufficiently low to prevent its becoming pasty, it will be first converted into PbO (perhaps only into Pb_2O) and SO_2. PbS does not oxidize readily, hence SO_2 will form slowly; part of the SO_2 combines with the O of the air and forms SO_3 by catalysis, and this combines with PbO to form $PbSO_4$; if Pb_2O is present, the SO_3 first converts it into PbO. The suggestion of Rammelsberg[11] that some PbS is directly oxidized to $PbSO_4$, without passing through the stage of PbO, does not appear to be valid, as the reaction $MS+2O_2 \rightarrow MSO_4$ is not reversible.[12] Under ordinary conditions the product of a roast is a mixture of PbO and $PbSO_4$. Thus Plattner[13] in roasting pure galena obtained the proportion $5PbO:2PbSO_4$. The recent research of Dempwolff[14] shows that if a roast is carried on at a temperature below 450° C. there is formed mainly $PbSO_4$, and that with slow oxidation in the presence of a catalyzer the entire PbS may be converted into $PbSO_4$. On the other hand, the more the temperature rises above 450°, the greater will be the amount of PbO formed; if it is raised above 750°, the reaction between PbS

[1] *Metallurgie*, 1906, III, 441.
[2] *Loc. cit.*
[3] Dissertation, Darmstadt, 1914.
[4] Tilden, *Proc. Roy. Soc.*, 1907, LXXI, 220.
[5] Kiliani, *Berg. Hüttenm. Z.*, 1883, XLII, 235, 375.
[6] *Compt. rend.*, 1902, CXXXIV, 1224.
[7] *Op. cit.*, 1902, CXXXV, 456, 734.
[8] Schmehl, *op. cit.*
[9] *Metallurgie*, 1909, VI, 170.
[10] *Op. cit.*, p. 145.
[11] Percy-Rammelsberg, "Die Metallurgie des Bleies," Vieweg, Brunswick, 1872, p. 39.
[12] Vondráček, *Oest. Zt. Berg. Hüttenw.*, 1906, LIV, 437.
[13] *Loc. cit.*
[14] *Metall. u. Erz.*, 1914, XI, 619.

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and $PbSO_4$ (§41) will begin, and the Pb set free will be oxidized by cupellation. The experiments prove the generally accepted rule that slow roasting at a low temperature produces more $PbSO_4$ than if the operation is carried on quickly at a high temperature. This may be expressed by

$$2PbS+7O=PbO+PbSO_4+SO_2 \text{ (slow roast)}$$
$$3PbS+10O=2PbO+PbSO_4+2SO_2 \text{ (quick roast)}$$

The proportions of PbO and $PbSO_4$ formed are dependent not only upon the temperature and amount of rabbling, but are governed largely also by the character of the minerals associated with the galena in the ore. Thus Plattner[1] in roasting galena from Bleiberg, Carinthia, which contained a small amount of blende and pyrite, obtained in his product the ratio $PbO:PbSO_4$; and in admixing 50 per cent. pyrite this ratio was changed to $2PbO:3PbSO_4$. The presence of other sulphides is likely to increase the amount of $PbSO_4$ formed.

The study of Bannister[2] of the effects of gangue minerals is of special interest. The galena concentrate with which he carried on muffle-roasts contained Pb 78.56, S 11.79, SiO_2 2.20, Fe 2.24, Zn 1.98 per cent. Fig. 33 gives the temperature record of a roast of galena mixed with 20 per cent. lime. The dotted line *CD* represents a blank test, the full-drawn line *AB* the charged muffle. Curve *AB* shows a slight evolution of heat between 180° and 260° C., due to the oxidation of a small amount of pyrite; at 560° a considerable rise in temperature takes place accompanied by a glowing of the charge; the evolution of heat ceases after 730°, but is followed by a second rise at 745° which is not sufficient to cause a glow; at 800° a third rise occurs which causes the charge to glow and the temperature to rise rapidly to 900°. The first evolution of heat, now shown to take place at 560°, was noticed by Percy[3] and by Hutchings.[4] The composition of the charge, after the first glow, had changed from PbS 70.4, CaO 20.9, SiO_2 1.7, N.D. 7.0 per cent. to PbS 28.3, $PbSO_4$ 7.3, PbO 28.7, CaO 16.0, $CaSO_4$ 10.4, SiO_2 1.6, N.D. 7.7 per cent.; and after the second glow to PbS *nil.*, $PbSO_4$ *nil.*, PbO 56.0, CaO 6.2, $CaSO_4$ 29.5, SiO_2 1.5, N.D. 6.8 per cent. The first analysis shows that during the first glow some PbS has been converted into $PbSO_4$, and a considerable amount into PbO, also that some

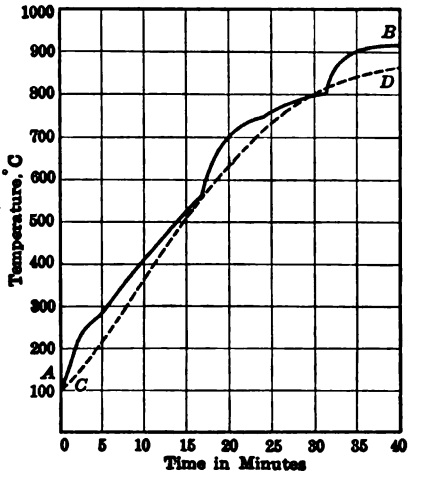

FIG. 33.—Muffle-roast, galena—lime mixture.

[1] *Loc. cit.*
[2] *Tr. Instit. Min. Met.*, 1912–13, XXI, 346.
[3] "Lead" p. 238.
[4] *Eng. Min. J.*, 1905, LXXX, 726.

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$CaSO_4$ has been formed; the second, that the remaining PbS and $PbSO_4$ have been converted into PbO, and the whole of the remaining S is present as $CaSO_4$. This proves that CaO has a decomposing effect upon PbS in an oxidizing roast; a fact intimated by Rivot,[1] and claimed also for BaO. Fig. 34 gives the temperature-curve for a mixture of galena with 20 per cent. SiO_2 having the composition Pb 72.6, SiO_2 21.7, N.D. 5.7 per cent. Between 180 and 290° there is again a slight evolution of heat due to the pyrite present; the first rise in temperature due to the oxidation of PbS occurs at 535°, the second at 730°, and the third at 810°. The first two evolutions take place at temperatures slightly lower, the third at a slightly higher temperature than do those noted with the lime-mixture; all three evolutions are less intense. After heating to 700° the composition of the charge has changed to PbS 30.0, $PbSO_4$ 26.7, PbO 17.8, SiO_2 21.2, N.D. 4.3 per cent.; and after 850° to PbS 1.0, $PbSO_4$ 51.1, PbO 22.7, SiO_2 22.0, N.D. 4.2 per cent. A comparison of the compositions of the products of the PbS–CaO and the PbS–SiO_2 mixtures shows that at these roasting temperatures CaO acts very differently from SiO_2, the latter serving only as a diluent of the charge. Practical experience with blast roasting has brought out the fact that charges with an admixture of CaO work better than those having other diluents

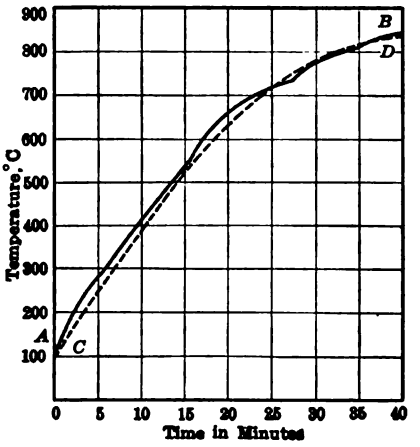

FIG. 34.—Muffle-roast, galena—silica mixture.

In the original paper, Bannister gives curves for mixtures of PbS with PbO–CaO, with $CaCO_3$, $CaSO_4$, MgO, Fe_2O_3, and $CaO+H_2O$. The research of Percy[2] upon the changes galena undergoes in the English reverberatory furnace supplements the laboratory work of Bannister.

The decomposition of PbS by a metal which has a stronger affinity for S than Pb, studied in 1833 by Fournet[3] has been reinvestigated by Schütz[4] who places the metals

Mn–Cu–Ni–Fe–Sn–Zn–Pb

in the given order, Mn forming the strongest and Pb the weakest sulphide. In the smelting of lead ores Mn need not be considered, as it practically always occurs as MnO_2, which enters the slag and is found only to a very small extent in matte. Of the other metals, Fe is the most important; the reaction $PbS+Fe=Pb+FeS$ forms the basis of what is called the *Precipitation Process*.

[1] "Traité de Métallurgie," Dunod, Paris, 1872, II, p. 42.
[2] "Lead," 234–240.
[3] Hofman, "General Metallurgy," 1918, 74.
[4] *Metallurgie*, 1907, IV, 659, 694.

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Practical experience has shown that the decomposition of PbS is never complete; and Schütz has proved that the reaction $MS+N=M+NS$ is in part reversible. In furnace work it is customary to calculate the Fe necessary for the decomposition of PbS according to the formula given: if less Fe is added, the resulting matte remains too rich in Pb; if an excess is given, it is wasted, as the Fe is simply dissolved by the PbS–FeS matte. The amount of Fe the matte can hold in solution varies with the temperature. The alloy series FeS–Fe has been studied by Friedrich,[1] the series FeS–Fe–PbS awaits investigation. An excess of Fe in the charge may be disadvantageous in the decomposition of argentiferous galena, since on account of the affinity of Ag_2S and FeS (see Matte, §180) more Ag will go into the matte than can be accounted for by the amount of Pb present. In addition to having the correct amount of Fe present to decompose PbS, the temperature of the reaction is of decided importance; the higher it is, within reasonable limits, the better will be the decomposition. A basic ferrous silicate ($4FeO.SiO_2$) will decompose PbS readily; the singulo-silicate ($2FeO.SiO_2$) shows little effect. In practice, the amount of Fe in excess of that required to flux the SiO_2 is available for the decomposition of PbS.

According to Berthier,[2] CaO and BaO have a decomposing effect upon PbS, if air is excluded and C present: $2PbS+CaO+C=Pb+(PbS.CaS)+CO$. The research of Schütz[3] shows that the presence of CaS in a charge greatly diminishes the yield of Pb, and that the harmful effect of BaS, while similar, is very much weaker.

40. Lead Sulphate, $PbSO_4$ (68.3 per cent. Pb; $207Pb+32S+64O=303PbSO_4+215{,}700$ cal.).—This occurs as anglesite; is formed in roasting PbS and in precipitating lead salts with H_2SO_4. The melting-point lies above 1100° C., but is difficult to determine on account of dissociation (see below). It is a poor conductor of electricity;[4] is slightly soluble in H_2O, less so in dilute H_2SO_4; somewhat soluble in concentrated H_2SO_4, in HNO_3 and solutions of nitrates, in hot HCl; in solutions of $Na_2S_2O_3$, the solubility increasing with concentration and temperature;[5] in $CaCl_2$, NaCl,[6] $Ca(C_2H_3O_2)_2$, and NH_3-salts.

$PbSO_4$ is decomposed in part by heating. The constitutional diagram $PbSO_4$–PbO by Schenck-Rassbach[7] is given in Fig. 35. It shows the existence of two stable basic sulphates, $PbSO_4.PbO$ with 42.39 per cent. PbO, and $PbSO_4.2PbO$ with 59.49 per cent. PbO; a third basic sulphate, $PbSO_4.3PbO$, with 68.78 per cent. PbO, is stable only below 880°, as it splits above this temperature into $PbSO_4.2PbO$ and PbO. There are also three eutectic mixtures,

[1] Hofman, "General Metallurgy," 213.

[2] "Traité des Essais, etc." II, p. 673; also Percy, "Lead," 55.

[3] *Loc. cit.*

[4] Kiliani, *Berg. Hüttenm. Z.*, 1883, XLII, 235, 375.

[5] Stetefeldt, C. A., "The Lixiviation of Silver Ores with Hyposulphite Solutions," New York, 1895, 25.

[6] *Met. Chem. Eng.*, 1916, XIV, 31.

[7] *Ber. deutsch. chem. Gesell.*, 1907, XL, 2185, 2947; 1908, XLI, 2917.

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PbO–$PbSO_4$.3PbO with 87 per cent. PbO, freezing at 820°; $PbSO_4$.2PbO–$PbSO_4$.-PbO with 53 per cent. PbO, freezing at 940°; and $PbSO_4$–PbO.$PbSO_4$ with 30 per cent. PbO, freezing at 950° C. In the solid state there are two transformations, one of pure $PbSO_4$ at 850°, the other of $PbSO_4$.2PbO at 450° C. In heating $PbSO_4$ in a current of air, Doeltz–Graumann[1] found dissociation to begin at 900° C., and to increase to 46 per cent. by heating 1.5 hr. at 1,000°. Hofman–Wanjukow[2] found that in a current of air $PbSO_4$ shows the first signs of dissociation at 637° and that energetic decomposition begins at 705° with the formation of $6PbO.5SO_3$. This salt undergoes a transformation at 847°, begins to be decomposed at 888°, to sinter at 896°, and to fuse at 910°; rapid discsciation into $2PbO.SO_3$ begins at 952° and is accompanied by volatilization of PbO. The later investigation of Mostowitsch[3] shows that partial dissociation begins only at 800°; that the speed of decomposition is slow up to 950° and quick when the temperature is raised above 950°; that fusion occurs between 950° and 1000° after a partial decomposition of the normal $PbSO_4$.

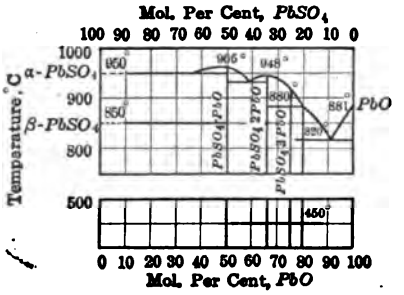

FIG. 35.—Constitutional diagram $PbSO_4$–PbO.

SiO_2 readily decomposes $PbSO_4$ at 1,030° C.[4] forming lead silicate, while the SO_3 driven off is split into SO_2 and O. Mostowitsch[5] has shown that in the presence of SiO_2 the dissociation temperature is not lowered; that the decomposing effect of SiO_2 is not proportional to the amount present, but rather that the reverse holds true, as decomposition is governed by the viscosity of lead silicates formed, the viscous slag enveloping $PbSO_4$ and retarding the decomposing action; and that the most rapid decomposition accompanied by the lowest loss of lead by volatilization lies between the singulo- and bi-silicate containing from 10 to 15 per cent. SiO_2.

The $PbSO_4$ obtained in roasting a siliceous galena is thus decomposed, if the temperature is sufficiently raised and the percentage of SiO_2 correctly apportioned. The operation goes by the name of *Slag-roasting*.

Fe_2O_3 also has a decomposing effect. Proske[6] found that it assists decomposition at 900° C. and higher temperatures, that its action is less energetic than that of SiO_2, and that at 1100° the entire SO_3 is driven off. He also found that Fe_2O_3 heated to 1200° is a more powerful decomposing agent than the ordinary red oxide. He further ascertained that if FeO in contact

[1] *Metallurgie*, 1906, III, 44.
[2] *Tr. A. I. M. E.*, 1912, XLIII, 523.
[3] *Tr. A. I. M. E.*, 1916, LV, 741; *Met. Chem. Eng.*, 1916, XV, 46.
[4] Friedrich, *Stahl u. Eisen*, 1911, XXXI, 2040.
[5] *Loc. cit.*
[6] *Metall. u. Erz.*, 1913, X, 415; *Eng. Min. J.*, 1913, XCVI, 207.

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with $PbSO_4$ is heated in a current of air, it is oxidized up to 800° by the O of the air, whereas above 800° it robs the $PbSO_4$ of some of its O.

Metallic Pb is oxidized by $PbSO_4$ at a high temperature, $Pb+PbSO_4=2PbO+SO_2$; Fe is oxidized and sulphurized, $4Fe+PbSO_4=Fe_3O_4+FeS+Pb$; CaO is sulphatized, $CaO+PbSO_4=CaSO_4+PbO$.

The reduction of $PbSO_4$ by C and CO to PbS is not as simple as is usually supposed because there are formed Pb and SO_2 besides PbS.[1] Reduction by C begins at 550° C. and is accompanied by the liberation of SO_2 and Pb, the PbS formed acting upon unreduced $PbSO_4$; the speed of reduction is quicker at 630° than at 550° and is finished at 700° with the two residual products PbS and Pb. Reduction by CO begins at 600°, also with the liberation of Pb and SO_2; it is very active at 630°. The double reaction noted furnishes an explanation for the elimination of S as SO_2 in the reducing fusion of a lead blast furnace; in fact, in smelting the gray slag from the ore-hearth as much as 50 per cent. of the S is usually expelled. The sulphate-S of $CaSO_4$ in blast-roasted ore has also little influence upon the matte-fall, as $CaSO_4$ is either reduced at 800° C. to CaS, which enters the slag as sulphide, or is dissociated by SiO_2 at 1000° C., when the resulting CaO follows the same course.

Furnace men in this country have found that part of the S of the $CaSO_4$ enters the matte; the same is the case with some English plants.[2] The whole question has become less important since double treatment in blast roasting has reduced the total S to 2 per cent. and less.

41. Reactions between Lead Sulphide, Oxide and Sulphate.—If intimate mixtures of PbS with PbO or of PbS with $PbSO_4$ in correct proportions are brought to a temperature at which softening begins, *i.e.*, a dull- to a light-red, reactions take place which liberate Pb and SO_2.

1. $PbS+2PbO=3Pb+SO_2-52{,}540$ cal.
2. $PbS+PbSO_4=2Pb+2SO_2-97{,}380$ cal.

If there is an excess of PbS or PbO over that called for by equation, the excess remains unaltered. If there is an excess of PbS in equation 2, the PbS remains unaltered; if there is an excess of $PbSO_4$, part or all of the lead is obtained as PbO.

3. $PbS+3PbO=3Pb+PbO+SO_2$.
4. $2PbS+2PbO=3Pb+PbS+SO_2$.
5. $PbS+2PbSO_4=Pb+2PbO+3SO_2$.
6. $PbS+3PbSO_4=4PbO+4SO_2$.

According to Lodin[3] the reaction of equation 1 takes place at 720° C., that of equation 2 at 670°, and that of equation 6 at 820°. Equations 1 and 2 form the basis of the *Roasting and Reaction* or *Air-reduction Process* (§55).

Schenck and Rassbach[4] have investigated the chemical processes that are

[1] Mostowitsch, *loc. cit.*

[2] Hutchings, W. H., letter, Aug. 4, 1916.

[3] *Compt. rend.*, 1895, CXX, 164.

[4] *Ber. deutsch. Chem. Gesell.*, 1907, XL, 2185, 2947; 1908, XLI, 2917; *Metallurgie*, 1907, IV, 455.

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possible by the interaction of Pb, PbS, PbO, $PbSO_4$, and SO_2. Under the guidance of the phase rule they may be grouped to form four equations.

(*a*) $PbS+2PbO \rightleftarrows 3Pb+SO_2$. This equation is reversible. Equilibria were found at 692° C. with a tension of SO_2 of 6 mm. Hg.; at 755° with 38 mm.; at 800° with 99 mm.; at 847 with 544 mm.; at 870° with 830 mm. The action of PbS upon PbO begins at 650°–660° C. and becomes very decided at 700°; at 800° PbS begins to become volatile, and the volatilization increases rapidly with the temperature.

(*b*) $PbS+PbSO_4 \rightleftarrows 2Pb+2SO_2$. This equation is also reversible. Equilibria were found at 609° C. with a tension of SO_2 of 30 mm. Hg.; at 655° with 155 mm.; at 700° with 422 mm.; at 723° with 735 mm. The action of PbS upon $PbSO_4$ begins at 550° C. and increases in velocity with the temperature, but is governed by the pressure of SO_2.

(*c*) and (*d*) $PbS+3PbSO_4 \rightarrow 4PbO+4SO_2$ and $Pb+PbSO_4 \rightarrow 2PbO+SO_2$. The equations show no equilibria even up to a pressure of SO_2 of 760 mm. Hg. The components begin to act at 550° C., and the reactions proceed from left to right irreversibly.

It will be noticed that the reactions were studied under pressures of pure SO_2 ranging from 6 to 830 mm. Hg. In furnace practice there is no pressure of SO_2; on the contrary there is a slight vacuum due to draft. In a furnace-charge, also, one has to deal not with pure SO_2 as in the experiments, but with air charged with small amounts, not over 7 per cent., of CO_2 and SO_2, the partial pressure of which is low. Lastly, SO_2, the gaseous phase, being withdrawn from the furnace as soon as formed, we have to deal with three instead of four components; hence the presupposed equilibria and reversible reactions have no existence, and equations (*a*) and (*b*) in furnace-work proceed only in one, the usual direction from left to right. As to reaction (*c*) with reference to blast roasting (§84), the forced draft carries off the SO_2 set free as soon as formed, and thus assists in the quickness of the roast besides increasing the temperature, both of which favor an acceleration of the process.

According to Reinders,[1] PbS acting upon $PbSO_4$ forms the basic sulphate $4PbO.PbSO_4$ which acting upon PbS liberates Pb, *viz.*, $PbS+7PbSO_4=4PbO.PbSO_4+4SO_2$ and $6\,PbS+4PbO.PbSO_4=14Pb+10SO_2$. If the two equations are added, and the intermediary product is omitted, there is obtained $7PbS+7PbSO_4=14Pb+14SO_2$, or $PbS+PbSO_4=2Pb+2SO_2$.

[1] *Zt. anorg. Chem.*, 1915, XXIII, 213; *Eng. Min. J.*, 1916, CII, 870.

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CHAPTER VI

LEAD ORES

42. Introduction.—Many minerals contain lead, but only two or three are found in sufficient quantity to be considered as important sources of lead. According to the character of the lead mineral, the ores of lead are divided into the two classes of sulphide ores (galena) and oxide ores (anglesite, cerussite, etc.) commonly called carbonate ores; both are made more or less impure by other metallic compounds and foreign matter. Viewing ores from the manner in which they are to be fluxed in the blast furnace, they are classed as siliceous, basic, and neutral ores, according as they require a base or silica, or are self-fluxing; relative to the necessity or desirability of eliminating most of the S before smelting, they are considered as roasting or non-roasting ores, practical designations which are in most cases synonymous with the terms of sulphide and oxide ores.

43. Lead Minerals—*Galena* (PbS; 86.6 per cent. Pb, 13.4 per cent. S).—This mineral is found well-crystallized in cubes, sometimes also in isometric octahedrons. Crystals are not so often found isolated as in irregular bunches. It occurs also in coarsely crystalline to fine granular varieties; crypto-crystalline galena is rare. Galena is found in most of the geological formations. It is usually mixed with more or less vein matter, such as limestone, dolomite, sandstone, quartz, earthy carbonates, barite, clay-slate, granite, gneiss, etc., which often have to be removed by a mechanical washing process before sending the mineral to the smeltery. The mine which works on the lowest grade galena ore is probably that at Mechernich, Rhenish Prussia, where small nodules of galena the size of a pea occur in a soft Triassic sandstone, the grains of which, about the size of millet, are slightly cemented by a clay or lime bond. The ore contains only 2.5 per cent. of galena, and the galena runs only about .6 oz. silver per ton.

Table 10 shows the rocks and formations in which some well-known galena deposits occur and the tenor of the ore before and after dressing.

In Tables 11 and 12 are assembled analyses of galena concentrates from Southeast Missouri.

The average lead content of the sulphide lead ores mined in the U. S. in 1914 was 5.7 per cent.,[1] that of lead-zinc ores 4.7 per cent.

[1] *Min. Res. U. S.*, 1914, I, 423.

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TABLE 10.—SULPHIDE LEAD ORES

Locality	Nature of rock	Geological formation	Raw ore, per cent. Pb	Dressed Ore		Reference
				Per cent. Pb	Ounces Ag per ton	
Mineral Point, Wis.	Dolomite	Ordovician	.....	.....	3.00	1
Rockville, Wis.	Dolomite	Ordovician	.....	.....	0.30	1
Granby, Mo.	Dolomite	Lower Carboniferous	.....	84	1.25	1
St. Joseph, Mo.	Dolomite	Cambrian	7	70	.....	2
North of England	Limestone	Carboniferous	8.5	70–77	8.00	3
Bleiberg, Carinthia	Limestone	Triassic	8	71	0.05	4
Přibram, Bohemia	Graywacke	Lower Silurian	20	37–38	76.50	5
Freiberg, Saxony	Gneiss	Archæan	3	18–70	17–88	6
Tarnowitz, Silesia	Dolomite	Triassic	6	75.5	13.50	7
Upper Harz, Prussia	Graywacke slate	Lower Carboniferous	9	64	25.00	8
Mechernich, Prussia	Sandstone	Triassic	2	56–60	3–4	9
Kellogg, Idaho	Limestone	Algonkian	11	60	30.00	10

1. "Geological Survey of Wisconsin," 1873–79, IV, 382.
2. Desloge, *Tr. A. I. M. E.*, 1889–90, XVIII, 262.
3. Hunt, R., "British Mining," Crosby Lockwood & Co., London, 1884, 899.
   Phillips, J. A., "Elements of Metallurgy," Lippincott, Philadelphia, 1887, 566.
4. *Oesterr. Zt. Berg. Hüttenw.*, 1890, XXXVIII, 286.
5. *Ibid.*, 1888, XXXVI, 567; and *Oesterr. Jahrb.*, 1891, XXXIX, 10.
6. "Freiberg. Berg und Hüttenwesen," Craz and Gerlach, Freiberg, 1883, 83.
7. *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1884, XXXII, 392.
8. *Ibid.*, 1880, XXX, 131, and Private notes, 1890.
9. "Bergbau und Hüttenbetrieb von Mechernich," Cologne, 1886.
10. Private notes.

TABLE 11.—ANALYSES OF GALENA CONCENTRATES FROM SOUTHEAST MISSOURI

Ag (*a*)	Cu	Pb	SiO_2	FeO	CaO	MgO	Zn	S	Ni	Co
1.30	0.127	68.6	1.4	5.1	3.1	1.4	0.8	15.5	0.06	.....
0.70	0.129	73.2	1.0	4.7	2.6	0.8	0.4	15.0	0.05	.....
1.10	0.248	71.6	1.1	5.2	2.3	0.9	0.6	15.6	0.10	.....
0.65	0.282	64.6	7.0	6.4	2.9	1.3	0.8	13.7	0.16	0.04

(*a*) Ounces per ton.

TABLE 12.—ANALYSES OF ORES AS MINED IN SOUTHEAST MISSOURI AND OF VARIOUS GRADES OF RESULTING CONCENTRATES (*a*)

	Ag (*b*)	Cu	Pb	SiO_2	Fe	Al_2O_3	CaO	MgO	Zn	S	Ni and Co
Ore	0.12	0.06	5.7	5.0	4.1	4.9	25.5	14.2	0.8	2.0	....
High-grade concentrates	0.7	0.13	73.2	1.0	3.5	.....	2.6	0.8	0.4	15.0	0.05
Medium concentrates	1.3	0.12	68.6	1.4	4.6	.....	3.1	1.4	0.8	15.5	0.06
Low-grade concentrates	1.0	0.30	65.8	0.5	3.1	0.5	4.3	2.8	1.7	13.7	....
Flotation slimes	....	0.50	45.0	9.6	4.4	3.1	7.5	4.2	4.0	12.3	....
Flotation slimes, high-grade	3.7	0.05	57.8	6.0	2.7	.....	2.2	1.4	9.4	15.5	....
Joplin concentrates	....	....	80.2	1.1	1.0	.....	0.4	.....	1.7	13.3	....

(*a*) Garlichs, A., *Bull. A. I. M. E.*, July, 1917, 1043.
(*b*) Ounces per ton.

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Galena often occurs in a very pure state, but is more generally mixed with other metallic sulphides. These are either pyrite, arsenopyrite, chalcopyrite, blende, bournonite, jamesonite,[1] etc., which occur as associated minerals; or silver, copper, zinc, iron, nickel, etc., which form chemical compounds with the lead sulphide. The associated minerals can usually be separated mechanically from the galena, but not always. Sometimes the admixture is too intimate, and then causes trouble and loss in the metallurgical treatment.

Galena is almost always argentiferous. The silver is rarely present in the native state; in galena poor in silver, it may appear as isomorphous silver sulphide, but a more common mode of occurrence is as a finely disseminated argentite or other rich silver mineral. Nissen and Hoyt[2] found that the limit of solid solution of Ag_2S in PbS is 0.2 per cent. and that the eutectic is not developed in concentrations below 2.7 per cent. The difference of form is important in connection with wet concentration. If the silver occurs as isomorphous sulphide, the loss in concentration will correspond, approximately, to the percentage of lead in the tailing; if as associated mineral, *e.g.*, tetrahedrite, it will be great, as this mineral, being very brittle, is readily crushed to a fine powder, and, being also lighter than galena, is carried off on the water. If the dark scum that is often seen floating on the water of jigs treating argentiferous galena be assayed, the main source of loss in silver will appear.

The tenor of silver in galena ores varies a great deal. The galena from Bleiberg, Carinthia, with 0.05 oz. silver per ton, represents probably the lowest amount, and occasional specimens from Idaho and Schemnitz, Hungary, with 2,042 oz. the highest.

It has often been said, and may sometimes still be heard, that coarse-grained galena is poor in silver, whereas fine granular varieties give higher assays; but Malaguti and Durocher[3] disproved it over sixty years ago. The minerals usually associated with galena—such as pyrite, blende, etc.—do not generally contain as much silver as the galena.

Percy[4] states that gold is as invariably present in galena as is silver, but it does not often occur in appreciable quantities. Nevins[5] records the occurrence of a galena ore in Sonora with 268.3 oz. Ag and 15.2 oz. Au per ton, in the concentrate of which no native gold could be found with the microscope. Metallographic methods, however, might have discovered it.

44. Anglesite ($PbSO_4$; PbO, 73.6; SO_3, 26.4; Pb, 68.3 per cent.); **Cerussite** ($PbCO_3$; PbO, 83.5; CO_2, 16.5; Pb, 77.5 per cent.).—Carbonate ores, using that term in a general technical sense as embracing all oxide lead ores, occur often in the form of a sand or an earth, bearing the name of sand or soft carbonates. In

[1] Smelting of: Ives-Ossa., *Eng. Min. J.*, 1909, LXXXVII, 891.
Holin, *Met. Chem. Eng.*, 1910, VIII, 686.
Raymond, *Eng. Min. J.*, 1915, XCIX, 9.
[2] *Econ. Geol.*, 1915, IX, 172.
[3] *Ann. Min.*, 1850, XVII, 85.
[4] "Lead," 96.
[5] *Eng. Min. J.*, 1905, LXXX, 769.

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other cases the particles of carbonate are cemented together by clay, iron, manganese, or silicious matter, forming compact amorphous or crystalline lumps called hard carbonates.

The minerals are seldom found as originally deposited; the form and composition are more likely to have been caused by chemical changes that have taken place since the galena, from which they resulted, was deposited. The sulphate usually formed by the oxidation of the sulphide is in most cases the compound from which the carbonate has resulted by the action of circulating waters holding alkaline or earthy carbonates in solution.

Anglesite as an ore is rare, as it is not often that galena is exposed only to the oxidizing action of air. It often occurs, however, with cerussite and represents the transition between the sulphide and the carbonate. This is further illustrated by pieces of galena found in carbonate ores, which contain a rind of anglesite with a surface changed to cerussite. Thus these two minerals will be found always near the outcrops of galena deposits. To what extent the decomposition has progressed depends on local circumstances.

The richness in lead of carbonate ores varies a great deal. If galena alone has been oxidized, the ores are likely to be rich; if the decomposing action attacked also the country rock, this may contaminate the ore to such an extent as to reduce the percentage of lead below the limit where it pays to treat the ore. The grade of carbonate ore is not so often raised by wet concentration as that of galena ore, as the losses in lead, and especially in silver, which passes off in the slimes, are almost unavoidable. In some instances the carbonate ore has been first leached with sodium hyposulphite to remove as much silver as possible, and then the lead concentrated in the wet way to a rich product. This was formerly done at the Old Telegraph Mine, Utah. Another method that has been tried is Krom's system of dry concentration, the result being a high-grade smelting ore, and a tailing and dust, to be treated in the wet way. Table 13 shows some very pure carbonate ores from Missouri resulting from correspondingly pure galena, and others from Colorado, contaminated with country rock. More recent analyses of carbonate ore are given later.

TABLE 13.—CARBONATE LEAD ORES

Locality	Per cent. Pb	Ounces Ag per ton	Chemist	Reference
South-Western Missouri..	72	.....	Chauvenet	Broadhead, "Geological Survey of Missouri," 1874, 710.
Granby, Mo.....	65	.....	Williams	*Tr. A. I. M. E.*, 1876–77, v, 315.
Leadville, Colo.....	38	25	Ricketts	*Tr. A. I. M. E.*, 1885–86, xiv, 287.
Leadville, Colo.....	21	65	Rolker	*Tr. A. I. M. E.*, 1885–86, xiv, 287.
Red Mountain, Colo.....	17	128	Kedzie	*Tr. A. I. M. E.*, 1887–88, xvi, 581.

The associated minerals undergo a process of oxidation with the galena and are generally found again in part in the carbonate ore, although, being more

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soluble than the lead sulphate, they may have been carried away entirely. The silver in oxidized ores is present mostly in the form of chloride (Leadville chlorobromide, bromide; and Broken Hill iodide) although it also occurs in its original form as sulphide and antimonide or arsenide. There is less likelihood of a uniform ratio between lead and silver in a carbonate ore than there is in a sulphide ore, as lead sulphate and carbonate show a behavior with solvents different from that of silver sulphide, chloride, and antimonide. Thus enrichment and impoverishment both in lead and silver can be accounted for easily.

45. Other Lead Minerals.—The following six oxide lead minerals occur often in carbonate deposits, but not in sufficient amounts to constitute an ore:

Pyromorphite, $PbCl_2+3Pb_3P_2O_8$; 76.4 per cent. Pb. Calcium fluoride often replaces in part the lead chloride; calcium takes the place of the lead combined with phosphoric acid; and arsenic acid that of the phosphoric acid.

Mimetite, $PbCl_2+3Pb_3As_2O_8$; 69.6 per cent. Pb. In the lead arsenate the lead is sometimes in part replaced by calcium, and the arsenic usually in part by phosphorus.

Vanadinite, $PbCl_2+3Pb_3V_2O_8$; 65 per cent. Pb.[1]

Crocoite, $PbCrO_4$; 63.9 per cent. Pb.

Wulfenite, $PbMo_4$; 57 per cent. Pb.

Stolzite, PbW_4; 45.4 per cent. Pb.

There might still be mentioned about 20 lead-bearing sulpharsenites, sulphantimonites, and sulphobismuthites,[2] sulphoarsenates, sulphantimonates,[3] etc. which occur in lead deposits, but they are only mineralogical curiosities.[4]

46. Lead Ores of the United States.—Lead ores occur in many parts of the world. The mines of Spain and Germany furnish the bulk of the European product, but it is not the present purpose to deal with that branch of the subject.

The occurrence of lead ores in the United States is best discussed under four heads: The Atlantic Coast; the Mississippi Valley; the Rocky Mountains; and the Pacific Coast.

47. Atlantic Coast.—The lead ores of the Atlantic coast occur in New York, New England, Virginia, North Carolina, and Tennessee. Those of New York and New England, which were worked in former times, are practically abandoned now, and those of the Southern States are worked only on a small scale. Virginia has some deposits of galena and blende with carbonate and silicate of zinc; the lead, however, is subordinate to the zinc.

[1] Treatment: Herrenschmidt, *Compt. rend.*, 1904, CXXXIX, 635; *Oesterr. Zt. Berg. Hüttenw.*, 1905, LIII, 201.
Larsh, *Eng. Min. J.*, 1911, XCI, 1248; 1913, XCVI, 1103.
Grider, *Min. Sc. Press*, 1916, CXIII, 389.

[2] Treatment: Bretherton, *Eng. Min. J.*, 1910, LXXXIX, 773.

[3] Treatment: Ives-Ossa., *Eng. Min. J.*, 1909, LXXXVII, 891.
Hahn, *Met. Chem. Eng.*, 1910, VIII, 686.
Raymond, *Eng. Min. J.*, 1915, XCIX, 9.

[4] Dana, "System of Mineralogy," Wiley, New York, 1898, 109–151.

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48. Mississippi Valley.[1]—This heading covers two divisions: the lead region of the Upper Mississippi Valley and that of Missouri.

Lead ore is mined in Oklahoma, Kansas, and Kentucky, but the output is small.

Upper Mississippi Valley.—In the southwestern part of Wisconsin occur the lead deposits which extend a small distance into the adjoining States of Iowa and Illinois. They are principally centered around Mineral Point and Platteville, Wis.; Galena, Ill.; and Dubuque, Iowa. Most of the lead produced comes from Wisconsin.

The ore is a non-argentiferous galena; it occurs in wholly undisturbed dolomitic limestone of the Trenton period in vertical crevices, flat crevices, or as an impregnation. The galena ("mineral") from the upper beds is pure and rich. As depth is gained, the associated minerals marcasite and blende increase in quantity and often predominate over the galena. Chalcopyrite is scarce, and is found more with pyrite and blende than with galena. Secondary minerals are not of frequent occurrence. Calcite and barite occur in the lower beds. The absence of nickel, cobalt, and arsenic is to be noted.

Mines of Missouri.[2]—In both the southeastern and the southwestern districts the ore is a coarsely crystalline galena, practically free from silver.

In the southeastern district, galena occurs disseminated through strata of dolomitic limestone of the Cambrian, lying almost horizontally. The ore as mined runs about 3.5 per cent. Pb and is concentrated to a product of 67 per cent. Pb. An analysis of a carload of concentrate[3] from the Bonne Terre and Flat River districts gave Pb 62.10, Cu 0.61, Fe 3.33, CaO 5.85, MgO 3.08, SiO_2 1.39 per cent. and Ag 1.3 oz. per ton. The ratio between CaO and MgO appears frequently to be about 2:1. Other analyses have been given in Tables 11 and 12.

Associated with the galena occurs pyrite containing nickel and cobalt. Chalcopyrite and nickel and cobalt sulphides with traces of arsenic are found in the Mine La Motte and Bonne Terre mines. The absence of blende is to be noted. The pyrite-bearing galena ores are concentrated separately from the pure galena to a product called "sulphide," which has the composition given in Table 14.

TABLE 14.—SULPHIDE CONCENTRATE OF SOUTHEASTERN MISSOURI

Locality	Pb	Fe	Ni, Co	S	Insol.	Chemist
Bonne Terre, 1884.....	21.86	16.21	0.61	.....	.....	Setz.
Mine La Motte, 1881...	17.37	.....	4.77	.....	.....	Neill.
Mine La Motte, 1881...	13.34	44.84	4.07	20.37	3.58	Neill.

[1] Van Hise, C. R., Bain, H. F., *Tr. Inst. Min. Eng.*, 1901–1902, XXIII, 370–434.

[2] Winslow, A., "Lead and Zinc Deposits," Missouri Geological Survey, Jefferson City, 1894.

Finlay, *Eng. Min. J.*, 1908, LXXXVI, 605.

[3] Editor, *Eng. Min. J.*, 1902, LXXII, 582.

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The Cambrian dolomitic limestone in which the ores occur contains about 3 per cent. SiO_2; the presence of barite is noted in some of the Upper Cambrian beds above the normal ore-horizon.

In the region of southwestern Missouri, which reaches into Kansas, Arkansas, and Oklahoma, lead and zinc ores are worked especially around Joplin.[1] The galena occurs in cherty limestone of the Lower Carboniferous or Mississippian period. When broken, it often emits a bituminous odor. It occurs in single or loosely aggregated crystals, also in crystalline masses of small dimensions imbedded in the limestone and in the beds of chert, the fragments of which are cemented together by a bluish-gray, clayey mass. Cadmium-bearing blende occurs plentifully in two forms, a coarsely crystalline and a granular variety; pyrite is subordinate. To be noted is the absence of chalcopyrite and barite. Dolomite and calcite are found with the galena, and often occur in coarsely crystalline masses. Concentrates contain from 74 to 85 per cent. Pb; the average for 1914–15[2] was 76–77 per cent. Pb. The analyses of Table 15 by Spencer[3] show the character of the concentrate shipped to smelting works:

TABLE 15.—GALENA CONCENTRATE FROM SOUTHWESTERN MISSOURI

	Henrietta	St. Louis and Aurora	Coffman	Stevenson & Wamper	Magnet	Ashcroft
H_2O, hygr.	0.114	0.111	0.140	0.014	3.728	0.922
H_2O, comb.	0.407	0.706	0.183	.....	.....	.....
Insol.	1.050	0.912	1.201	.....	.....	.....
SiO_2	.....	.....	.....	0.903	1.257	0.208
S	13.948	14.537	12.364	14.040	14.351	13.722
As	.....	.....	.....	0.109	0.099	0.027
Zn	1.748	3.246	5.720	3.359	1.045	0.482
Pb	81.693	80.059	79.751	80.225	77.255	83.537
Fe	0.357	0.294	0.251	0.117	1.066	0.640
Al_2O_3	0.238	0.260	0.239	0.345	0.411	0.153
CaO	.....	.....	.....	0.396	0.228	0.003
CO_2	.....	.....	.....	0.316	0.184	0.002
	99.555	100.125	99.849	99.824	99.624	99.696

More recent figures are those of Petraeus and Waring,[4] who give Pb 78.4 per cent. as an average of one hundred shipments, and Pb 70–84.4, Fe 2.24, and Zn 1.78 per cent. as an average of fourteen shipments.

49. Rocky Mountains.—To this division belong the occurrences of argentiferous lead ores in Colorado, Montana, and New Mexico.

[1] Smith, W. S. T., *Bull.* 213, U. S. Geol. Survey, 1903, 217.
[2] Waring, J. W., *Priv. Com.*, October, 1915.
[3] *Bull. Missouri Mining Club*, 1, No. 1, 30; No. 2, 51.
[4] *Eng. Min. J.*, 1905, LXXX, 721.

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Colorado.—The deposits of argentiferous lead ores of Colorado used to be the most important of the country, but the production of the State has fallen off greatly, furnishing in 1915 only 32,000 tons of lead.

The San Juan region, situated in the southwestern part of the State, is characterized by immense quartz veins traversing older and younger eruptive rocks. In the neighborhood of Rico (Dolores County) occur silver lead ores mainly in the form of blankets between carboniferous limestone and eruptive rocks; in the Red Mountain district (Ouray County) deposits are found filling cavities in silicified andesite. The minerals are argentiferous galena, silver-bearing tetrahedrite, pyrargyrite, and native silver; bismuth-silver minerals are frequent; small amounts of gold are found; blende occurs in considerable quantity. The gangue is quartz with kaolinite and rhodocrosite; barite is common; fluorite also occurs. In the blanket deposits of Rico oxides of iron and manganese are prominent. Near the outcrops the ores are often completely changed into sand carbonates. In 1914 Dolores County furnished about 250 tons lead, and Ouray County 1,055; in the latter the Sneffels district is the leading producer.

Lake County.—This is still one of the important lead producers of the State, having furnished in 1914 over 5,000 tons of lead. Of the deposits in sedimentary rocks those of Leadville are still in the lead. The principal ore used to be argentiferous galena with its secondary products of decomposition—anglesite, cerussite, and pyromorphite. The accessory minerals were blende and its secondary products, carbonate and silicate of zinc. The gangue consisted of silica, free and combined, iron and manganese, and various clays charged with the same metals. Such ores are still mined, but form only 6 per cent. of the product of the district. Of the oxide ores there prevail silicate and carbonate of zinc to an extent of 21 per cent. Sulphide ores, encountered with increase of depth of mine, form at present 58 per cent. of the product; of these, 21 per cent. are blende with pyrite and some galena, and 37 per cent. pyrite with lead and some copper.

Chaffee County.—This used to be an important producer of oxide silver-lead ores occurring in limestone. The lead produced in 1914 amounted to 700 tons which came mainly from mines working sulphide ore, although some carbonate is still produced. Of the sulphide ore, some is galena; the bulk, however, is a mixture of blende, galena, pyrite, and chalcopyrite. The Chalk Creek and Monarch districts contain the principal mines.

Pitkin County.—The leading district is that of Aspen which came into prominence in 1884. The ores occur in the same geological horizon as the Leadville deposits, but are not found at the contact of limestone and eruptive rock; they occur in the highly folded and faulted limestone as impregnations of fine-grained argentiferous galena in limestone along fault fissures. The district in 1914 produced 12,000 tons of lead; this came mainly from sulphide concentrates, as the ore when mined contains 9–10 per cent. Pb and 3 oz. Ag per ton present mainly as polybasite. Associated with lead are zinc minerals. Table 16 gives analyses of some Colorado ores delivered at present to smelteries.

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TABLE 16.—ORE SHIPMENTS FROM COLORADO

Locality	Ounces per ton		Per cent.								
	Au	Ag	Pb	Cu	Zn	S	Insol.	SiO_2	Fe (Mn)	CaO	$BaSO_4$
San Juan region	0.26	28.1	37.1	0.9	1.4	9.0	48.8	40.1	4.2	0.6	.....
San Juan region	0.70	2.7	7.6	7.2	8.1	25.7	20.0	16.3	22.3	0.3	.....
San Miguel region	1.49	29.9	10.5	0.9	....	n.d.	39.8	26.4	20.9	0.5	.....
San Miguel region	10.04	28.6	10.1	0.1	1.3	n.d.	1.2	0.8	13.6	.....	.....
Lake County	0.02	5.0	28.9	...	....	0.02	12.9	11.6	24.3	.....	.....
Lake County	0.31	3.7	23.4	0.1	....	0.7	.....	37.9	15.2	0.2	.....
Lake County	0.06	8.0	9.4	...	....	0.6	48.1	41.6	18.0	0.6	.....
Lake County	0.51	7.4	18.8	...	14.9	n.d.	4.3	.....	23.5	.....	.....
Chaffee County	2.38	10.6	54.2	0.6	6.9	n.d.	1.8	1.5	10.8	.....	.....
Chaffee County	0.01	1.8	9.3	0.3	1.9	0.3	10.7	9.6	41.4	.....	.....
Pitkin County	.....	7.6	14.2	0.1	5.0	2.2	.....	10.1	4.0	17.7	1.1
Pitkin County	.....	4.0	33.0	0.5	2.6	8.3	.....	9.1	3.7	13.0	.....

Montana.—The argentiferous lead ores of Montana are insignficant in comparison with those coming from the great copper deposits. The production of lead in 1914 was nearly 5,000 tons; of this, 56 per cent. came from the mines in Silver Bow, 10 per cent. from Cascade, and 9 per cent. from Lewis and Clark Counties. The ore, occurring as replacement veins in gneiss and igneous rocks, is a mixture of silver-bearing galena and blende, which has to be concentrated. Lead concentrates of 1914 shipped to smelteries averaged 16.27 per cent. Pb, and lead-copper concentrates 23.26 per cent. Pb; the blende concentrate averaged 2.43 per cent. Pb.

New Mexico.—The lead production of this State appears to be decreasing, having fallen from 1,980 tons in 1913 to 886 in 1914. The Magdalena, Grant and Cooks districts are the leading producers. The deposits of Magdalena Mountains occur in Palæozoic limestone which has been cut by granite porphyry dikes; the oxidized zones contain lead, silver, and zinc; the unaltered zone much blende with some galena and chalcopyrite. The deposits of the Grant and the Cooks districts form fissure veins in porphyry and granite. Table 17 gives analyses of some ore shipments.

TABLE 17.—ORE SHIPMENTS FROM NEW MEXICO

Ounces per ton		Per cent.								
Au	Ag	Pb	Cu	Zn	S	Insol.	SiO_2	Mn Fe	CaO	Al_2O_3
0.12	6.2	16.9	.....	.....	1.6	.....	26.4	27.1	2.7	1.6
0.29	24.1	12.8	6.6	2.1	2.7	.....	34.0	11.8	3.6	.....
.....	5.7	40.0	.....	1.2	0.2	.....	28.2	4.0	0.6	.....
.....	3.2	24.2	1.2	5.3	0.8	.....	32.5	11.8	0.8	.....
0.02	4.8	26.0	.....	1.9	1.8	35.4	30.0	12.6	1.0	.....

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50. The Pacific Coast.—The argentiferous lead ores under this heading occur in Nevada, Utah, Idaho, Arizona, and California.

Nevada.—The production of argentiferous lead in Nevada has greatly diminished, amounting in 1916 to 112,000 tons. The ore used to come from the Richmond and Eureka Consolidated mines in the Eureka district, which at present furnishes only about 100 tons lead. The leading mines are situated in Lincoln, Clark, and White Pine Counties with 54, 38, and 8 per cent. of the State's product; the deposits form veins in quartzite, limestone, and slate. In Lincoln County the Pioche district is the most important, the ores are carbonate and sulphide; in Clark County the Yellow Pine district furnishes zinc-lead sulphides to concentrating plants; in White Pine County the small amounts of lead ore come mainly from the Cherry Creek district, with mines in the neighborhood of the smelter town McGill. The ore is mainly sulphide, only slightly altered to anglesite and cerussite. Table 18 gives analyses of lead ore shipments.

TABLE 18.—ORE SHIPMENTS FROM NEVADA

Ounces per ton		Per cent.								
Au	Ag	Pb	Cu	Zn	S	SiO_2	Fe	Mn	CaO	Speiss
0.005	2.4	2.9	10.0	1.8	0.4	11.8	34.7	12.7	2.5	.....
0.005	4.0	0.0	3.43	tr.	0.5	9.8	23.7	.....	20.9	3.6
0.005	12.5	39.5	tr.	tr.	2.0	15.6	8.8	.....	3.8	.....
tr.	17.6	9.7	4.64	7.2	0.4	35.7	13.3	.....	5.2	5.7

Utah.—The lead production of this State is of great importance, having attained in 1914 the large figure of 86,000 tons. The argentiferous lead deposits form more or less regular bodies in limestone or at the contact of limestone and eruptive rock. The ores are carbonate with anglesite strongly prevailing; the secondary minerals extend downward to considerable depths; small amounts of galena occur. Many of the mines have been worked down to the sulphide zone. The leading mines of the State are in the Bingham or West Mountain district. The ores form an irregularly mineralized zone between limestone and quartzite, and sometimes porphyry. The limestone and porphyry are more or less decomposed near the deposits. The lead ore is carbonate containing more or less silver, ferruginous clay, and some galena. The siliceous ore is an oxidized siliceous pyrite. With increase of depth, the ores of the district have changed to sulphide, and most of them must be concentrated.

In the Tintic district occur, as replacement deposits in limestone, heavy galena and carbonate which change into silicious ore running high in lead and copper; most of them go direct to smelteries.

The Park City region of Summit county has oxide and sulphide lead ores as fissures penetrating folded and faulted limestone, quartzite, and shales; the larger part of the production comes from beds in limestone. Most of the ores have to be enriched by concentration.

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Tooele County, with the Ophir and Rush Valley districts, is prominent producers of oxide and sulphide lead ore in the ratio of 3 : 1.

In Beaver County is the celebrated Horn Silver mine which is still a producer. The ore, sulphide with some carbonate, comes mainly from replacement deposits in limestone, some from contact deposits.

Table 19 gives analyses of ore received by smelteries.

TABLE 19.—ORE SHIPMENTS FROM UTAH

District	Ounces per ton		Per cent.								
	Au	Ag	Pb	Cu	Zn	S	Insol.	SiO_2	Fe	CaO	Speiss
Bingham	0.14	3.9	4.4	0.51	3.7	19.2	.....	45.0	15.6	2.7	.....
Bingham	0.23	8.0	17.1	0.4	4.8	37.8	.....	5.2	29.2	1.0	.....
Bingham	0.049	3.6	13.3	0.6	3.8	20.8	30.6	.....	16.3	4.9	.....
Bingham	0.166	8.7	14.5	1.1	7.8	24.6	23.4	.....	17.4	3.0	.....
Little Cottonwood	0.03	14.6	6.6	2.33	4.4	0.5	.....	28.6	9.3	11.6	.....
Ophir	0.005	11.1	10.9	2.0	6.5	17.0	22.5	.....	16.7	3.0	.....
Park City	0.048	31.3	19.7	1.57	12.0	13.9	31.0	.....	8.10	3.0	.....
Tintic	0.156	17.5	13.3	0.1	5.5	....	53.8	.....	5.3	3.4	.....
Tintic	tr.	22.8	18.0	0.4	4.0	2.1	.....	40.6	2.7	3.6	.....
Tintic	tr.	4.2	32.0	tr.	7.2	3.2	.....	30.0	2.1	4.6	.....
Tintic	0.10	34.7	13.1	0.22	2.8	0.7	.....	65.2	7.1	0.7	.....
Tintic	0.085	14.7	18.4	tr.	4.5	0.7	.....	52.6	14.0	1.7	.....
Tintic	0.05	25.2	8.4	0.51	1.2	1.7	.....	58.7	12.6	1.0	1.4

Idaho.—The argentiferous lead ores are the most important of the country, the State having produced 170,000 tons of lead in 1916. The mines worked at present lie in four districts, the Coeur d'Alene (Shoshone County), the Texas (Lemhi County), the Birch Creek (Fremont County), and the Mineral Hill (Blaine County). The deposits of Coeur d'Alene are fissure veins of varying

TABLE 20.—COEUR D'ALENE LEAD-SILVER ORES[1]

Mine	Ore		Concentration ratio	Concentrates	
	Pb, per cent.	Ag, ounces per ton		Pb, per cent.	Ag, ounces per ton
Bunker Hill and Sullivan	13.5	3.9	4–5:1	55	20
Gold Hunter	3 to 5	4	11:1	50	40
Hecla	9.2	5.8	5.5:1	50	30
Helena	9.2	5.8	5.5:1	47	28
Hercules	15	3 to 10	4:1	50	45
Last Chance	10	4	5:1	50	20
Morning	8	3	7:1	50	16
Standard-Mammoth	6	5	8.5:1	50	41

[1] Rowe, *Min. World*, 1909, XXX, 428.

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thickness in Algonkian sedimentary rocks, such as quartzite, shale, and schist, which have been much folded. The ore is galena and galena-blende in the ratio of 3 : 1, and has to be concentrated. The average assay of the lead ores shows 9 per cent. Pb and 4 oz. Ag per ton, of the lead-zinc ores 6.5 per cent. Pb and 6 oz. Ag per ton; the lead concentrates shipped to smelters average 52.46 per cent. Pb and 29.19 oz. Ag per ton.

Table 20 gives the tenor in Pb and Ag of ores as mined, the rate of concentration, and the values of Pb and Ag of the concentrates, of some of the leading mines.

Table 21 gives analyses of some concentrates received by smelteries in 1916.

TABLE 21.—ORE SHIPMENTS FROM COEUR D'ALENE, IDAHO

Ounces per ton		Per cent.									
Au	Ag	Pb	Cu	Zn	S	Insol.	SiO_2	Fe	Mn	CaO	Al_2O_3
0.004	29.72	49.5	.....	2.50	9.60	21.10	.....	6.90	.....	.....	.....
.....	38.68	46.7	.....	0.80	8.20	23.20	.....	8.50	.....	.....	.....
0.005	19.88	50.5	.....	2.40	11.26	28.10	.....	4.30	.....	.....	.....
0.03	12.80	26.5	0.25	5.80	0.50	.....	.....	13.50	.....	4.0	.....
0.004	29.72	49.5	.....	2.50	9.60	21.10	.....	6.90	.....	.....	.....
tr.	35.00	45.0	0.50	5.50	12.50	.....	8.5	17.00	2.00	0.5	2.00
.....	20.3	60.4	.....	0.90	10.30	4.0	3.7	9.80	.....	0.4	.....
.....	13.0	35.9	.....	3.20	8.2	20.4	17.2	14.2	.....	0.3	.....
.....	20.4	51.5	.....	2.60	10.9	7.6	6.9	11.2	.....	0.2	.....
.....	33.5	48.9	0.09	3.10	11.5	15.4	13.9	8.0	.....	0.7	.....
0.009	17.26	37.2	.....	9.90	13.10	17.10	.....	10.90	.....	.....	.....
0.03	21.20	55.0	0.30	7.30	14.60	.....	8.0	8.80	.....	0.2	.....
.....	25.00	37.0	0.20	10.50	13.00	.....	12.0	16.00	1.00	0.5	2.00
.....	24.8	56.7	.....	6.30	10.6	3.9	3.3	8.10	.....	.....	.....

Arizona and California.—The product of Arizona in 1916 was 15,000 tons. The leading district is the Warren or Bisbee (Cochise County). The ores are mainly sulphide. The average assay of the ore shipped in 1914 was Pb 18.57, Cu 0.03 per cent.; Ag 13.92, Au 0.08 oz. per ton.

The product of California in 1916 was a little over 3600 tons lead. Most of the lead came from Inyo county, some from Shasta County.

51. Purchasing Lead-Silver Ores.[1]—The treatment of a lead-silver ore is both a metallurgical and commercial operation; hence the process used and the

[1] Kirchoff, *Min. Res. U. S.*, 1885, 281.
Furman, *School Min. Quart.*, 1893–94, XV, 1.
Austin, *Eng. Min. J.*, 1907, LXXXIII, 226; *Min. Sc. Press*, 1914, LIX, 170.
Barbour, *Eng. Min. J.*, 1911, XCII, 364.
Caetani, *Min. Sc. Press*, 1913, CVI, 684, 724; *Min. Ind.*, 1913, XXII, 877.
Turnbull, *op. cit.*, 1916, CXIII, 133.
Fulton, C. R., "The Buying and Selling of Ores and Metallurgical Products," Technical Paper 83, Bureau of Mines, 1915;.
"Report of Smelter and Ore Sales Investigation Committee, State of Colorado, 1917."

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manner of carrying it out must be not only correct physically and chemically, but the result obtained must make an adequate return in recovery. The price paid a mine operator for a lead ore will have to cover at least the value of the net yield of metal, the total cost of treatment, and the profit it is desired to make. This total cost of treatment includes the net cost, the cost of transportation of metal to the refiner, the cost of refining, the selling expense, and the transportation of refined metals to the consumer. There must also be made a deduction from the market quotation of the metal in the price paid for the metal in order to furnish a protection against loss from the variation of market prices in the large stocks that have to be carried in the form of ore, and intermediary or finished product.

In purchasing, the character of lead-bearing mineral and the composition of the gangue have to be considered as well as the quantity of ore that is offered. If the lead mineral is a sulphide, the ore generally will have to be roasted or blastroasted; if it is a carbonate, this is not necessary. The gangue may be either self-fluxing, acid or basic, *i.e.*, require no fluxes, require a base (FeO, MnO, CaO), or require an acid (SiO_2) to form a desirable slag. Most ores are silicious, hence basic ores are likely to command a higher price than acid. In purchasing basic ores, the base paid for is Fe with its substitute Mn; often CaO also receives some consideration. The so-called "base excess" is that amount of available Fe and Mn which is obtained by adding the percentage of Fe to that of Mn, and deducting the SiO_2, or $Fe+Mn-SiO_2=$Base Excess.

The impurities in an ore affect its price. They may necessitate a preliminary roasting (S, As), may impair the fusibility of the slag (ZnO, MgO, Al_2O_3), or may cause loss of Pb and Ag by slagging or volatilization (Zn, As, Sb), or finally may render the lead impure (Zn, As, Sb, Cu). The price paid for an ore will therefore be in inverse ratio to the percentage of impurities present.

The quantity of ore offered at a time will influence the price that can be paid, as the larger the amount, the lower is the general and the special cost, provided of course that the quantity offered is not excessive.

A smeltery located in a railroad center draws its supplies from a number of mining districts and can work more cheaply than one which has been erected in a mining district furnishing only a single class of ore. Such a central plant treats not only argentiferous lead ores, but extracts the precious metal also from real silver ores, called dry ores on account of their lack of lead, by mixing them with ores that contain more lead than is required for the charge. Smelting can thus often compete with milling on account of the higher percentage of precious metal extracted from the ore.

In bidding for an ore, deductions are made for actual and possible loss, and for the total cost of smelting, which will vary with the causes mentioned.

The value of lead is given in per cent. or, what is the same, in units of 20 lb. to the ton of 2000 lb. The percentage is determined by the dry or wet assay. As the latter gives figures which are from 1–1.5 per cent. higher than

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the former, a deduction of this amount is sometimes made from the wet assay to arrive at the dry assay, which is the basis of settlement. In Utah a combination method is in use which determines in the wet way the lead-content of the button. Assays ought to agree within 0.5–1.0 per cent. The prices paid are regulated not simply by the fluctuation of the market, but also largely by the scarcity or abundance of available lead ore at the works. No payment is made for lead if present in amounts of less than 5 per cent.

There are in operation three methods for the payment of lead ores.[1]

1. *Payment by the unit and quotation*, a system customary in Colorado. The method is based upon what is called a "quotation" of $4.00 per 100 lb. lead. The "quotation" is not the sales price in New York, but is figured from the latter. In making a settlement there is added or deducted 1 ct. per unit for every rise or fall of 5 cts. in the quotation.

If the sales price in New York is $4.00, the quotation is 90 per cent. of this price, or $3.60. If the sales price in New York lies between $4.00 and $4.50, the quotation is obtained by adding to $3.60 one-half of the excess of the sales price over $4.00. Supposing the sales price in New York to be $4.30 per 100 lb., the quotation will be $3.60+½ ($4.30−$4.00)=$3.75. If the sales price in New York is higher than $4.50, the quotation is obtained by adding to $3.60 one-half of the excess of the sales price between $4.00 and $4.50, and all of the excess above $4.50. Supposing the sales price in New York to be $5.00 per 100 lb., the quotation will be $3.60+½ ($4.50−$4.00)+($5.00−$4.50) =$4.35.

In Table 22 are given the amounts paid per unit of lead in ore on the basis of a $4.00 quotation.

TABLE 22.—PAYMENT FOR LEAD IN ORE ON BASIS OF $4.00 QUOTATION

Lead content, dry assay, per cent.	Price per unit	Lead content, dry assay, per cent.	Price per unit
5–10	$0.35	25–30	$0.49
10–15	0.40	30–35	0.51
15–20	0.45	over 35	0.51
20–25	0.47		

In ascertaining the value of lead in an ore containing 35 per cent. Pb, with the sales price in New York of $4.30 per 100 lb., we start with the price per unit in the table, which is $0.51. From this item have to be deducted as many cents as there are 5-ct. units in the difference between $4.00 and the calculated quotation of $3.75. This difference is 25, or five 5-ct. units, hence a deduction of 5 cts. per unit is made from $0.51, the figure of the table, which leaves $0.46 to be paid per unit. The amount to be paid per unit of lead for the same ore, with the sales price in New York of $5.00 per 100 lb., is with the quotation figures above: $0.51+⅕ ($4.35−$4.00)=$0.58.

[1] Fulton, *op. cit.*, 23.

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2. *Payment for 90 per cent. of the lead content at the sales price, with a deduction per pound*, a system customary in Utah. The sales price is the average New York price for the week preceding the day of the transaction; the deduction is 1.5 cts. per lb. if the combination assay shows less than 25 per cent. Pb; it is 1.25 cts. if the assay gives more than this. An ore contains 36.5 per cent. Pb wet (or 35 per cent. by the combination assay); the sales price in New York is $4.30 per 100 lb. The payment for the lead-content of a ton is 90 per cent. of 700 lb. at the rate of 4.30 minus 1.25 cts. or 3.05 cts., which equals $19.21. In reality the ore assaying 36.5 per cent. Pb contains 730 lb. lead, which at the rate of 4.3 cts. per pound has a gross value of $31.39. The difference between this and the amount received is $12.18.

3. *Payment for 90 per cent. of the lead content at 90–95 per cent. of the sales price*, a system common with Idaho and Mississippi Valley ores. A ton of ore with 36.5 per cent. Pb wet or 35.0 per cent. Pb dry and a sales price in New York of $4.30 per 100 lb., will bring 90 per cent. of 700 lb. lead at 90 per cent. of $4.30 per 100 lb., or $24.38. The gross value based on the wet assay would be $31.39.

The payments made under the three methods for an ore containing 35 per cent. Pb dry with a sales price in New York of $4.30 per 100 lb. have given the values $15.29, $12.18, and $7.01; the treatment charges correct the discrepancies so that the miner receives about the same money by whatever method the lead may have been valued.

In some plants the treatment-charge is expressed in an additional deduction of 1.25 cts. per lb. lead instead of in a given number of dollars per ton. This amounts to $25.00 a ton. Austin[1] calculates that for Salt Lake City the total treatment including freight, refining, etc. is $18.00 per ton of lead.[2]

The value of *copper* is given in per cent.; assays should agree to 0.25–0.50 per cent. The amount is usually computed as being 1.0–1.3 per cent. less than the wet assay; hence no payment is made for ores assaying less than 1.3 per cent. The price paid is that of the average New York quotation of electrolytic copper for the week previous to the day of the transaction, less 4.5–5 cts. per lb. to cover the cost of freight and refining.

The value for *silver* and *gold* is given in ounces troy per ton of 2000 lb. Assays for Ag are required to check within 0.5–1.0 oz. per ton, and of Au within 0.01–0.03 oz. The price paid for silver is 95 per cent. of the New York quotation, the deduction of 5 per cent. being intended to cover the loss in smelting; no payment is made if the ore contains less than 1 oz. per ton. Gold is paid for at $19.00–$20.00 per oz.; no payment is made for less than 0.03–0.05 oz. per ton. The standard value of gold is $20.67 per ounce. A smeltery receives $20.40 from the refinery.

There are various ways of arriving at the minimum smelting charge for an ore that is offered for purchase. One of them is to base the cost of smelting on what is called a neutral ore, and then to debit and credit the actual ore, for

[1] *Min. Sc. Press*, 1914, CIX, 170.

[2] As refiners have since raised their charges this figure is now from $20.00 to $22.00 per ton.

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which the smelting charge is to be fixed, as it differs in composition from the standard. This so-called neutral ore or neutral schedule varies with the districts and differs sometimes in the same locality. Thus Furman[1] gave it in 1893–94 for Denver, Colorado, as consisting of SiO_2 30, Fe 30, Pb 13, Zn 8, S 5 per cent., crediting for Fe, CaO and to some extent for MgO+BaO, and debiting for SiO_2. Austin[2] gave in 1914 for Utah the slag-composition SiO_2 33, FeO 33, CaO 20, ZnO 8 per cent. as the basis of calculation. A third and more general statement is to define a neutral ore as one in which the Insoluble Residue is equal to the sum of Fe and Mn it contains, and to make varying rates for Fe, CaO, MgO, BaO, ZnO, Al_2O_3, S, and As. At present some smelteries have abandoned, at least in their dealings with mine operators, the idea of a neutral ore; they give credits for Fe, Mn, CaO, debits for Insol. (SiO_2), S, Zn, As, Sb, Bi, and make supplementary charges for bricking and treatment. Following the last conception there are in operation at present the following scales of prices.

Iron is paid for at the rate of 5–10 cts. per unit, assays agreeing to within 1 per cent.

Lime is paid for at the rate of 2–5 cts. per unit; usually no payment is made unless there is present a considerable amount of CaO, say 10 per cent. and over; assays are required to agree within 1 per cent.

Insoluble is charged for at the rate of 10–12 cts. per unit; assays agreeing within 1–2 per cent.

Zinc has no charge against it as long as the amount present does not exceed 5 per cent., in some cases 10 per cent.; any excess is charged for at the rate of 30–50 cts. a unit; assays must agree within 1 per cent.

Sulphur is charged for at the rate of 20–25 cts. a unit to a maximum of $3.00 per ton of ore, the assumption being that this figure covers the cost of roasting an ore with 12 per cent. S and over; sometimes there is no penalty for 2 per cent. S.

Arsenic.—This is usually figured as speiss; up to 5 per cent. speiss there is no charge; any excess over this figure is charged at the rate of 20 cts. a unit. Some smelteries figure on As as such and charge 5 cts. per unit in excess of 1 per cent.

The basic charge for treatment is $1.00 per ton for an ore with 30 per cent. Pb; a credit of 5–10 cts. a unit is allowed for every unit in excess of 30 per cent., and debit of 10 cts. made for every unit below 30 per cent. This treatment charge is below cost, but is made up by not giving full price for lead. The rationale of these treatment charges, is given by Austin.[3]

The following example will illustrate the manner in which ores are usually paid for at present. The chemical analysis shows:

Insol. 40, Fe 10, CaO 5, Pb 20, Cu 3, Zn 8, S 5 per cent.; Ag 50 and Au 1 oz. per ton. The credits and debits will give the net value per ton.

[1] *Loc. cit.*
[2] *Loc. cit.*
[3] *Min. Sc. Press*, 1914, CIX, 170.

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Credits	Per ton
Au, 1 oz. @ $19.00 per oz.	$19.00
Ag, 50 oz. @ 95 per cent. @ $0.50 per oz.	23.75
Pb, 20 per cent. (wet assay) minus 1.5 units = 18.5 per cent. = 370 lb. 370 lb. @ 90 per cent. = 333 lb. @ (5.0–1.5 cts.) =	11.66
Cu. 3 per cent. minus 1 unit = 2 per cent. = 40 lb.; 40 lb. @ (17–5 cts.) =	4.80
Fe, 10 per cent. @ 6 cts. per unit =	0.60
CaO, 5 per cent. @ 2 cts. per unit =	0.10
Total credit	$59.91

Debits	Per ton
Treatment	$1.00[1]
Lead, 30–20 = 10 per cent. @ 8 cts. per unit	0.80
Insoluble, 40 per cent. @ 12 cts. per unit	4.80
Sulphur, 5–2 (free) = 3 per cent. @ 25 cts.	0.75
Zinc, 8–5 (free) = 3 per cent. @ 30 cts.	0.90
Total debit	$8.25
Net to owner, $59.91–$8.25 =	$51.66

52. Purchasing Non-argentiferous Ores.[2]—In the Mississippi Valley lead concentrates used not to be sold and bought, at least officially, by assay but on a "flat" basis. The representative of the smeltery, the "buyer," went to the different mines, and, guided by previous experience in purchasing from the same mine, offered a certain amount per ton. This has been changed. In southwestern Missouri ores are sold on a market basis after they have been sampled[3] and their lead-contents determined by wet assay. The unit quantity of lead concentrate is 1,000 lb. and not the usual ton of 2,000 lb. Lead ores are paid for[4] at the price fixed in St. Louis once a week, usually Thursdays, for ore assaying 80 per cent. Pb wet; there is made a deduction of $0.50 a unit for every per cent. of lead below the 80-per cent. standard, there is paid a premium of $0.50 a unit for every per cent. in excess of it. Suppose a galena concentrate contains according to the seller's assay 75.30 per cent. Pb, and to the buyer's 75 per cent. The results are averaged, giving 75.15 per cent. Let the basic price for 80 per cent. be $25.00 per 1,000 lb. The difference, 80.00 − 75.15 = 4.85, multiplied by $0.50 = $2.43, is deducted from $25.00, and furnishes the price of $22.57 to be paid for 1,000 lb.

In southeast Missouri lead ores are sold by yearly contracts on a metal basis and not on a market basis as in southwestern Missouri. In Wisconsin[5]

[1] Since the heavy increase in New York quotations, this charge has been temporarily increased to $2.50 in many instances.

[2] Clerc, *Eng. Min. J.*, 1885, XL, 4.
Blake, *Tr. A. I. M. E.*, 1893, XXII, 564.
Spencer, *Bull. Mo. Mining Club*, 1, 1886, No. 2, p. 51.
Finlay, *Eng. Min. J.*, 1908, LXXXVI, 606.

[3] Editor, *Eng. Min. J.*, 1914, XCVIII, 900.

[4] Waring, W. G., Priv. comm., November, 1915.

[5] Wittich, *Eng. Min. J.*, 1915, C, 24.

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the manner of selling lead ores is similar to that of southwestern Missouri, Kansas and Oklahoma.

53. Metallurgical Treatment in General.—Metallic lead is extracted from its ores exclusively by smelting in carbon-heated furnaces. Electric furnaces have been proposed, especially for zinc-lead sulphides, but have not as yet come into use. Volatilization methods have been suggested.[1]

If the ores are free from silver, as, *e.g.*, in the Mississippi Valley, the resulting lead usually goes to market after it has been in part purified by liquating and poling. In a few instances, such lead is desilverized by the Parkes process, as the amount of impurity present is considerably reduced, and the higher price paid for the better grade of lead and the small amount of silver recovered make the operation profitable. In 1914[2] 15 per cent. of the lead from southeastern Missouri was desilverized. If the ore is argentiferous, the silver passes for the most part into the lead (lead bullion) which is then desilverized (Chapter X).

Wet and electrolytic methods have found no general application. Wet methods may be suited to complex zinc-lead ores when mechanical concentration has proved a failure in separating blende from galena.[3] The object of the various processes is to render the zinc soluble and furnish a lead-residue which can be smelted satisfactorily, as well as a zinc solution for the recovery of metal. Processes of this character belong to the metallurgy of zinc rather than to that of lead.

Of the numerous electrolytic processes proposed, only one, that of Salom,[4] has been in successful operation; it produced spongy lead to be used in storage batteries.

The smelting of lead ores is carried on in three types of furnaces, the reverberatory, the ore-hearth, and the blast furnace. Of these the first has become almost obsolete; the ore-hearth in recent years has become of considerable importance with a limited class of ores; the blast furnace is the leading apparatus, as it can treat quickly and economically all kinds of ores in large quantities.

[1] Wells, *Min. Sc. Press*, 1917, CXIV, 507.
Ralston-Williams-Udy-Holt, *Bull. A. I. M. E.*, August, 1917, p. 1205.
Larson, *Min. Sc. Press*, 1917, CXV, 275 (Bunker Hill).
[2] *Min. Res. U. S. Geol. Survey*, 1914, I, 100.
[3] *Met. Chem. Eng.*, 1916, XIV, 31.
[4] Richards, *Electrochem. Ind.*, 1902–03, I, 18.
[Salom, *Tr. Am. Electrochem. Soc.*, 1902, I, 87.

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CHAPTER VII

SMELTING LEAD ORES IN THE REVERBERATORY FURNACE

54. Introduction.—Smelting sulphide lead ores in the reverberatory furnace was very common until within the last few years, especially in England, Belgium[1] and Austria,[2] where suitable galena concentrates are available. Since the advent of blastroasting with subsequent blast furnace smelting, reverberatory work has become nearly obsolete. In the United States[3] there were in operation in Missouri in 1877 fifty-six reverberatory furnaces, and in 1914 only four. The furnace is doing very satisfactory work in the smelting of cyanide residues by the Taverner process.[4] This subject belongs, however, to the metallurgy of gold and silver, and is not treated here.

The discussion of reverberatory work on ores has been retained, with slight modifications, in the present edition, as the practice covers many metallurgical considerations of general interest and of direct value to the refiner of lead bullion in the working of his intermediary products.

55. Outline.—The process carried on in the reverberatory furnace is the *Roasting and Reaction* or *Air Reduction Process* which, as shown in §41, is based upon the interaction of PbS, PbO, and $PbSO_4$ at temperatures below 1,000° C. which sets free Pb and SO_2. Formerly the so-called *Precipitation Process*, based upon the decomposition of PbS by Fe (§39) was in operation with raw ores at Vienne and Poullaouen, France,[5] and Chicago;[6] also with roasted ores at Par and Point, Cornwall.[7] This practice has been given up on account of the high cost and the large loss of metal.

The roasting and reaction process in the reverberatory furnace requires two operations, one following closely upon the other, and both being repeated several times.

[1] *Min. Ind.*, 1901, X, 424.

[2] *Op. cit.*, 1906, XV, 533.

[3] Pulsifer, *Min. Eng. World*, 1914, XL, 1148.

[4] Taverner, *J. Chem. Met. Soc. S. Africa*, 1902–03, III, 70, 103, 121, 134; *Eng. Min. J.*, 1903, LXX, 75, 184.
Carter, *op. cit.*, 150.
Clevenger, *Tr. A. I. M. E.*, 1904, XXXIV, 891.
Rusden, *J. Chem. Met. Min. Soc. S. Africa*, 1905, V, 288; *Eng. Min. J.*, 1905, LXXX, 688.
Swinney, *Tr. Inst. Min. Met.*, 1906–07, XVI, 151; *Eng. Min. J.*, 1907, LXXXVIII, 608.
McGraw, *op. cit.*, 1912, XCIV, 840; 1914, XCVII, 606.
Clark-Sharwood, *Tr. Inst. Min. Met.*, 1912–13, XXII, 132.
Clark, *Tr. A. I. M. E.*, 1915, LII, 3.

[5] Kerl, B., "Grundriss der Metallhüttenkunde," Felix, Leipsic, 1881, p. 24.

[6] Jernegan, *Tr. A. I. M. E.*, 1873–74, II, 279.

[7] Percy, "Lead," 257.

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1. *Oxidation.*—The ore, crushed fine enough to pass a 4- or 5-mesh sieve, is spread in a layer 3–4 in. deep over the hearth of the furnace and is heated gradually to a dull-red heat (500° to 600° C.). The roasting is carried on in such a way that only a part of the PbS is converted into PbO and $PbSO_4$, the rest remaining undecomposed. The temperature at which this roasting is carried on and the time given to it depend on the character of the ore. Pure galena requires a low temperature to avoid agglommeration; if pyrite, blende, or calcite are present the roasting can be accelerated; the lower the temperature, the more sulphate will be formed. During the operation the fire on the grate is kept low and open, and the charge is raked frequently, to expose as much of the ore as possible to the action of air and heat and to prevent agglomerating, which obstructs oxidation.

2. *Reduction.*—The second operation is that of raising the temperature to about 800° C., so that the oxygen compounds may react on unchanged sulphide. The resulting Pb runs down the inclined hearth and collects in a basin, the SO_2 passes off through the flue, and the residue remains on the hearth. The temperature during the reduction period must be low, so that the charge may be only in a pasty condition, as the reactions are very imperfect if the ore is melted. By well filling up the grate with fuel the required temperature is obtained and unconsumed air excluded. The charge is stirred at intervals to bring sulphide and oxide constituents into intimate contact.

As it is not possible to roast a large amount of lead ore uniformly in one operation, the first reaction that takes place on raising the temperature will not extract all the lead. The resulting pasty residue will be rich PbS, consisting mainly of PbS with some PbO, $PbSO_4$, $PbSiO_{x4}$, and gangue. The temperature is lowered and air is admitted. A second roasting takes place and is followed by a second reaction. It takes several repetitions of the process to extract the bulk of the lead. With each one, the temperature must be slightly raised because the amount of lead diminishes. To counteract the melting of the charge, slaked lime is added, which acts mechanically by rendering the charge less fusible and more spongy; it may also assist the process chemically by liberating the lead and by decomposing the sulphide, thus helping the silver in the residue to pass into the lead. Toward the end of the process there will not be enough PbS left to react on the excess of $PbSO_4$ and PbO. To reduce these to PbS and Pb, and to make the charge more porous, coal is mixed in; then the roasting and reaction can proceed again.

Each successive operation will be of shorter duration than the preceding one and the lead each time a little less rich in silver. The first lead may contain four times as much silver as the last.[1]

56. Products.—The products of smelting in the reverberatory furnace are:

1. *Lead,* holding in suspension particles of ore and other solid matter, which

[1] Fallize, *Rev. Un. Min.*, 1859, VI, 421; *Berg. Hüttenm. Z.*, 1860, XIX, 359; 1863, XXII, 285. Bouhy, *Ann. Min.*, 1870, XVII, 159; *Berg. Hüttenm. Z.*, 1870, XXXII, 382; 1871, XXXIII, 52, 84, 92, 97, 150.
Teichmann, *Zt. Berg. Hütten. Sal. Wes. i. Pr.*, 1866, XIV, 232.

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are removed by liquating and poling. If the ore contains arsenic, antimony, or copper, some of these elements combine with the lead and have to be removed by refining (§241).

2. *Gray slag*, a more or less matted mixture of lead, lead sulphide, oxide, sulphate, silicate, gangue, cinders, and lime in varying proportions. Its tenor in lead and silver depends on the character of the ore and on the extent to which the residue has been treated in the furnace. In some cases it is crushed and washed to save only the metallic lead; in others, especially with silver-bearing ores, it is smelted in the blast furnace.

3. *Flue-dust*, composed of particles of unchanged or oxidized ore, of volatilized lead that has been converted into oxide, carbonate, and sulphate, and of fuel. If the ore contains blende, oxidized zinc compounds will also be found. The amount of flue-dust formed will vary with the temperature at which the roasting and reacting operations have been carried on, and also with the skill of the furnace-man in manipulating the furnace and the charge. As it consists principally of oxidized compounds, it is worked in with subsequent charges and shortens the time required for roasting. If very impure, *e.g.*, through arsenic and antimony, it is smelted in the blast furnace with the gray slag. The resulting lead is hard and has to be refined.

4. *Hearth bottom*, consisting of hearth material soaked to some depth with metal. It is worked up in the same manner as the residues.

57. Limitations.—The roasting and reaction process in the reverberatory furnace has the following advantages: the ore is treated in the raw state, the apparatus is inexpensive, inferior raw fuel is used, hardly any fluxes are required, the bulk of the metal in a pure state is quickly extracted at a low temperature with little loss by volatilization, and if the ore is argentiferous the larger part of the silver follows the lead, and only a small quantity is left in the residue, which is either thrown away or treated at a higher temperature in the blast furnace. The great disadvantage of the method is that it is very much limited by the character of the ore. To be suited for the reverberatory furnace an ore must be a rich galena or a mixture of galena with carbonate (the former prevailing), that does not contain less than 58 per cent. Pb, 70 per cent. being a common figure. It may not contain more than 4 or 5 per cent. SiO_2; and the non-silicious associated minerals, such as blende, pyrite, chalcopyrite, calcspar, and barite, may be present only in small quantities. The process requires much fuel and many hours of skilled labor per ton of ore treated.

58. Influence of Foreign Matter.—The quantity and quality of lead that can be obtained from a given lead ore will depend largely on the nature and proportion of the other constituents. These may be silica and argillaceous matter, oxides of iron, limestone (dolomite), barite, fluorspar, pyrite, chalcopyrite, blende, antimony, arsenic, silver (gold), and oxide lead ores.

Silica and argillaceous matter have an injurious influence in both stages of the process on account of their readiness to combine with lead oxide. It has been found by experiment that with more than 5 per cent. of SiO_2 an ore cannot be treated by the roasting and reaction process. But even such a small amount

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as 0.5 per cent. makes itself felt by coating particles of ore with the silicate that has been formed, thus preventing the action of the air during the first period and obstructing the reactions when the temperature is raised.

Oxides of Iron.—Siderite is sometimes found with galena ores, but most of it can be removed by dressing the ore before smelting. The small amounts which remain with the galena quickly lose their CO_2 (380°–415° C.) during roasting, and the Fe_3O_4 or Fe_2O_3 acts as a stiffening ingredient while the lead is being extracted.

Limestone (dolomite) acts on the whole advantageously during the entire process, as it hinders the fusing of the charge. Any chemical action it may have is so slight that it can be regarded as practically inert matter. It loses some of its CO_2 at about 800° C. and is to a small extent converted into $CaSO_4$. The investigations of Hofman and Mostowitsch[1] have shown that the reduction of $CaSO_4$ by C to CaS begins at 700° C. and is complete at 1000°, and that CO begins to act also at 700° C., but finishes the reduction at 850–900°. There may be formed therefore some CaS during the last and hottest stage of the process. This CaS can act to some extent upon $CaSO_4$ forming CaO and SO_2; but even at 1000° the reaction is incomplete because the necessary intimate contact of molecules is not attained even in a laboratory experiment. It may also be oxidized by the O of the air, which becomes active at 800° and ceases to be effective at about 950°, when a mixture of approximately 75 per cent. $CaSO_4$ and 25 per cent. CaO is formed. Most of the limestone will remain unchanged and like all indifferent substances will retard somewhat the roasting by preventing the air from having free access to the particles of galena and obstructing the reactions by interfering with the necessary intimate contact of sulphide and oxide. The highest allowable amount is 12 per cent.

Barite remains practically unchanged in the reverberatory. However, $BaSO_4$ may undergo changes similar to those of $CaSO_4$ and at lower temperatures. Mostowitsch[2] has shown that the reduction by C to BaS begins at 600° C. and is complete at 800°, and that the effect of CO starts at 650°, is 98 per cent. complete at 800°, and 100 per cent. at 1,050°.

Fluorspar, which melts at 1378° C., also remains unchanged unless[3] it combines with barite and $PbSO_4$ and thus increases the fusibility of the charge. Experimental evidence of these practical facts is wanting.

Pyrite, which loses 1 mol. of S at 700° C. is beneficial in the first stage. It favors the formation of lead sulphate and assists the oxidation of galena through SO_3 liberated by the decomposition of $FeSO_4$ or $Fe_2(SO_4)_3$. Small quantities of pyrite have also a favorable effect during the reaction period, the Fe_3O_4 or Fe_2O_3 making the charge less fusible. If present to a considerable amount, say from 10 to 12 per cent.,[4] too much iron sulphide remains in the charge

[1] *Tr. A. I. M. E.*, 1910, XLI, 763.
[2] *Metallurgie*, 1909, VI, 462; *Eng. Min. J.*, 1909, LXXXVIII, 601.
[3] Percy, "Lead," p. 488.
[4] Bouhy, *Ann. Min.*, 1870, XVII, p. 179.

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and is liable to form double compounds with the lead sulphide. With from 35 to 40 per cent. of pyrite[1] the reverberatory process can not be used.

Chalcopyrite has no special effect during the roasting, and behaves in a manner similar to that of iron pyrite in the reaction period, with this addition, that some of the copper enters the lead and impairs its quality.

Blende.—During the roasting period blende is partly converted into ZnO and $ZnSO_4$, and the latter somewhat decomposed, but much blende remains unaltered. From 4 to 5 per cent.[2] assists the roasting; 10 or 12 per cent. prolongs it greatly and reduces the output of lead; with from 20 to 24 per cent.[3] very little lead is obtained; and from 35 to 40 per cent. prohibits the process. The loss in silver with blende-bearing ores is principally a mechanical loss that takes place during the roasting period.

Antimony.—This occurs with some galena ores as a simple or multiple sulphide. It has a deleterious effect even if present in small quantities of 2 or 3 per cent. Stibnite[4] is readily fusible and causes caking of the ore. The Sb_2S_3 and the Sb_2O_3 are both volatile, thus causing loss. The oxide also combines readily with PbO forming xPbO.Sb_2O_3, which is retained to a great extent by the gray slag; further, Sb_2S_3 and Sb_2O_3 react upon each other, giving Sb, which is in part volatilized, causing further loss, and the remaining metal finally combines with the lead, making this hard. Thus antimony is probably the most harmful metal that can be associated with the lead.

Arsenic.—The pyrite found with lead ore sometimes contains arsenic. Next to antimony, arsenic is the worst enemy of the process. It causes losses by volatilization through the As_2O_3 formed in roasting; by slagging, through the combination of As_2O_5 with PbO forming yellow arsenate; and finally by the reduction to metal which combines with the lead and impairs its quality.

Silver and Gold.—Most of the silver of galena ores passes into the lead; gold behaves in a similar way.

Oxide lead ores, such as anglesite and cerussite, assist the operation, as they reduce the time of roasting.

59. Classification of Reverberatory Methods.—The practice of the roasting and reaction process varies in different smelting works. At some the principal stress is laid on extracting as much lead as possible in the reverberatory; while others aim to obtain only the major part of the lead in the reverberatory and to extract the rest from the rich residue by smelting it in the blast furnace, thus recovering a larger percentage of lead. Then, some establishments roast the ore slowly at a low temperature, which is advisable for the recovery of a large percentage of lead; while others hasten roasting by raising the temperature quickly to the permissible limit, the aim being to save labor and time, which is done at the expense of the lead. As regards the form and size of furnaces and the position of the lead-well there are also characteristic differences.

[1] *Ibid.*, p. 178.
[2] *Ibid.*
[3] Rivot, "Métallurgie du plomb," p. 46.
[4] Hofman-Blatchford, *Tr. A. I. M. E.*, 1916, LIV, 671.

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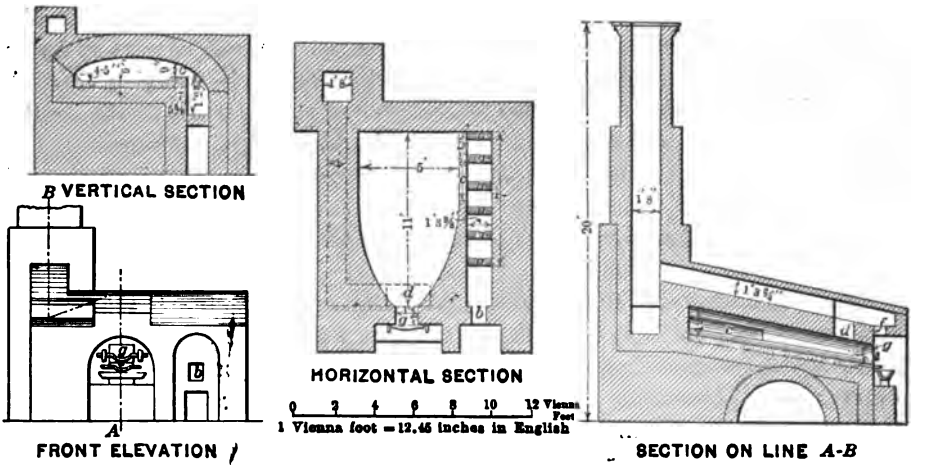
The reverberatory furnace practice may be classed under three main heads: 1. The Carinthian Method; 2. The English Method; 3. The Silesian Method.

60. The Carinthian Method.—The characteristics of this method are the smallness of the charge; the slow roasting, so that for every part of PbS one part of $PbSO_4$ and at least two of PbO are formed; the low temperature at which all the operations are carried on; and the aim to extract all the lead in the reverberatory. The hearth is inclined toward the flue and the lead is collected outside of the furnace.

(a). *Lead-smelting at Raibl,*[1] *Carinthia.*—The ore worked is galena (partly coarse, with from 72 to 75 per cent. Pb, and partly fine, with from 67 to 73 per cent. Pb); in exceptional cases the lead contents go as low as 58 per cent. The other constituents of the ore are anglesite, cerussite, wulfenite, blende, willemite, calcite, fluorite, and asbestos, but these form only a very small quantity. The following analyses by Philips[2] in 1845 show the composition of low-grade coarse and fine ore:

	PbS	$PbCO_3$	ZnS	Sb_2S_3	$CaCO_3$	Insol.	Total
Coarse.....	76.6	4.0	13.2	0.2	4.6	0.4	99.0
Fine.....	76.0	4.0	13.0	0.2	5.0	0.4	98.6


B VERTICAL SECTION
HORIZONTAL SECTION
0 2 4 6 8 10 12 Vienna Feet
1 Vienna foot = 12.45 inches in English
A
FRONT ELEVATION
SECTION ON LINE A-B

FIGS. 36 to 39.—Reverberatory furnace, Raibl, Carinthia.

The drawings (Figs. 36 to 39) of the furnace show an inclined hearth (9½°) with only one working door, *g*, below the flue, *d*. On the same side is the door, *b*, leading to the fireplace. The grate, slightly more inclined than the hearth, is parallel to the long axis of the furnace. It is built of stone and has six trans-

[1] Thum, *Berg. Hüttenm. Z.*, 1863, XXII, 196; "Official Report," *Oester. Zt. Berg. Hüttenw.*, 1889, XXXVII, 297; Report: "Bleivergiftungen," Hölder, Vienna, 1905, pt. 1, p. 18; *Min. Ind.*, 1906, XV, 535.

[2] *Ann. Min.*, 1845, VIII, p. 293.

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verse openings. The firebridge is at *c*; the opening, *f*, carries off any lead fumes into the main flue. The hearth terminates at its lower end in a cast-iron gutter, over which the lead runs into the mold. Fig. 37 shows the support for the working tools. The furnace is built of sandstone and ordinary red brick; the working bottom, which is renewed every 4 or 5 weeks, is made by tamping down firmly fireclay, probably a mixture of raw and burnt clay. It is made impervious to lead by glazing with gray slag. The heating-up of the furnace, leading to the fritting of the slag, is done slowly to prevent the cracking of the tamping. Furnaces are usually built in pairs, being placed side by side. They last from 5 to 6 years. The fuel used is cordwood, later replaced by lignite. Dimensions are given on the drawings and in §63.

The mode of operation is as follows: the furnace, barely red-hot from a previous charge, is repaired, if necessary, and the charge of about 400 lb. introduced through the working door, *g*, and spread out over the upper part of the hearth near the bridge to a thickness of from 1⅛ to 1½ in. No fresh fire is made, the heat of the furnace and the glowing fuel from the previous charge furnishing sufficient heat for the first slow roasting. The ore, containing a small amount of blende, is raked every quarter of an hour. With pure galena the raking is not repeated so frequently, as the quick oxidation would liberate sufficient heat to make the ore cake. The beginning of the caking is indicated by its adhering to the rake. The roasting period has come to an end when the blue sulphur flames disappear, drops of lead are seen near the front end of the hearth, and the ore feels sandy. The roasting period lasts 3 hr., during which time the charge is rabbled from eight to nine times and from five to six sticks of cordwood are consumed. The temperature is then raised and kept pretty uniform throughout the reaction which sets in. The attendant works his charge once every quarter or half hour, and raises the temperature slightly when the flow of lead ceases. This second period lasts from 5 to 6 hr., consumes from 16 to 18 sticks of wood, and furnishes the first half of the lead, which, on account of its freedom from impurities, is called virgin lead. The attendant now stops firing until he has collected the residue from all parts of the hearth into one heap. He then takes a few shovelfuls of ashes and breeze from the ash-pit, throws them on the heap of residue, and works them in in order to remove lead and to reduce lead oxide and sulphate. He then urges the fire as quickly as he can, and the so-called third period of the process, that of slag reduction, has begun. The further manipulations are the working of the residue and the stirring-in of breeze of charcoal, until after 3 hr. the rest of the lead, the "slag-lead," has been extracted. This has to be liquated before it can be marketed.

The practice is varied in some works by raking out the residue after the ashes and breeze have been stirred in and introducing a new charge. This is worked in the usual way. The residue from this second charge is not withdrawn, but that from the first added and both worked together for slag-lead. In this case the reduction of the slag occupies from 7 to 8 hr.

The final residue is withdrawn from the furnace and sorted into gray slag

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with 4 per cent. Pb, which is thrown away, and a product to be crushed and concentrated; the heads, which assay from 50 to 60 per cent. Pb, going back to the furnace in one of the subsequent residue-charges.

There is one furnace-man, working 24 hr., who has a helper during the day (12 hr.); later the work was done in 12-hr. shifts.

Tabulated results are given in §63. Of the products no satisfactory analyses exist except of the lead.

TABLE 23.—ANALYSES OF CARINTHIAN LEAD

	Virgin lead		Slag lead	
	a[1]	*b*[2]	*a*[1]	*b*[2]
	Per cent.	Per cent.	Per cent.	Per cent.
Cu	0.00069	0.00010	0.00075	0.00100
Ag	0.00025	0.00003	0.00025	0.00008
Fe	0.00055	0.00700	0.00088	0.00770
Ni	.....	.....	trace	.....
Zn	0.00076	.....	0.00082	.....
S	0.01476	0.00400	0.01785	0.00130
Sb	trace	0.05700	0.00703	0.14340
As	trace	.....	0.00721	0.01920

In 1893 the furnace and method were at last given up.[3] The Belgian reverberatory furnace is now in use and is worked by the Silesian method (see § 62).

(*b*) *Lead Smelting at Engis, Belgium.*[4]—The method of working at Engis differs in some respects from that at Raibl. The ores are pure. They contain, according to an average of several analyses: PbS 93.56, ZnS 3.74, FeS_2 2.31, $CaCO_3$ 0.35 per cent., and traces of silver. They are free from arsenic and antimony.

The furnace has the ordinary form of a reverberatory. The hearth, widest in the middle, is slightly contracted at the bridge and narrows down considerably at the flue. The furnace has two doors, one at the side and one beneath the flue, below which there is a small kettle with a separate fireplace to receive the lead. The fuel used is bituminous coal. The furnace bottom, oval in cross-section, is 2 ft. 8¾ in. thick at the top of the bridge, and is inclined toward the flue, so that no lead is collected in the furnace. At the flue the thickness is 1 ft. 5¾ in. The furnace has a brasque working bottom. The brasque consists of two parts by volume of ordinary brick clay and one of coke ground fine enough to pass a 4-mesh screen. Old bottoms, containing usually about 2 or 3 per cent. Pb, are ground up and mixed with

[1] *Oester. Jahrb.*, 1872, XXII, 389.

[2] *Op. cit.*, 1879, XXVII, 188.

[3] *Oester. Zt. Berg. Hüttenw.*, 1893, XLI, 283.

[4] Bouhy, *Ann. Min.*, 1870, XVII, 159; reprint, "La fabrication du plomb," Dunod, Paris, 1870; also in *Berg. Hüttenm. Z.*, 1870, XXXIX, 381; 1871, XL, 52 ff.

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new brasque. All the brasque needed for one furnace bottom (about 4000 lb. of clay and 1600 lb. of coke) is prepared by two men in 24 hr. In addition to the ordinary test for the correct amount of moisture (that the brasque, when squeezed in the hand, shall cohere into a lump, but not have sufficient moisture to adhere to the hand) another one is used, that of throwing with force a ball of brasque against the wall, to which it should adhere.

In tamping, a layer of brasque 8 in. thick is first spread out evenly on the brick hearth and tamped down to 1 in.; the second 8-in. layer is not rammed down so firmly, being reduced only to 2⅜ in.; the subsequent layers are made by using smaller amounts of brasque, as when spread out they are only about 4 or 5 in. thick. The bottom is not impervious to lead, which filters through to a slight extent, collects on top of the underlying red brick, and also passes into the joints. A bottom lasts about 6 weeks. It wears off quickly during the first 2 weeks, but then resists pretty well abrasion by the tools and corrosion by chemical action.

In starting a furnace with a new bottom the warming lasts 6 hr., slight cracks that may form are closed with brasque. Then a small charge of 220 lb. of low-grade ore with from 45 to 48 per cent. Pb is spread over the hearth and the temperature raised gradually for 5¼ hr., the ore being raked from eight to ten times. The brasque becomes superficially soaked with lead and coated with a mixture of various lead compounds more or less melted. There result from this charge 4 lb. of lead, 5 lb. of rich slag-like material, and 130 lb. of matte-like material containing 48 per cent. Pb. The charges are increased in weight, and richer ore is worked until after the third day the furnace can do normal work.

The method of working is the same as at Raibl, with the exception that all the lead is not extracted in the reverberatory. The residue, forming 12 per cent. of the original charge and assaying from 17 to 20 per cent. Pb, is smelted with puddle-cinder in a small blast furnace. One furnace-man with a helper works a charge in 12 hr. Tabulated dimensions and results are given in §63.

The subjoined analyses show the composition of the residue:

TABLE 24.—ANALYSIS OF RESIDUE

SiO_2	PbO	ZnO	$(FeAl)_2O_3$	CaO	S	Total
29.86	25.50	25.33	15.03	3.70	0.38	99.80
28.10	32.80	20.80	14.70	3.20	0.10	99.70

(*c*) *Lead Smelting in the Air Furnace.*[1]—The roasting and reaction process,

[1] Williams, "Geological Survey of Missouri: Industrial Report," 1877, pp. 8–101; *Tr. A. I. M. E.*, 1876–77, 314.

Broadhead, "Geological Survey of Missouri," 1874, p. 492.

Ingalls, W. R., "Lead and Zinc in the United States," McGraw-Hill Book Co., New York, 1908, pp. 63, 112 and foll.

Pulsifer, *Min. Eng. World*, 1914, XL, 820 (Wisc.), 1148 (Mo.).

The furnace has been used to a considerable extent in the Mississippi Valley, where it origi-

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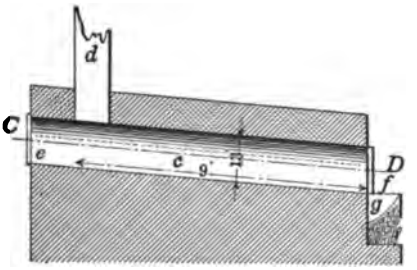
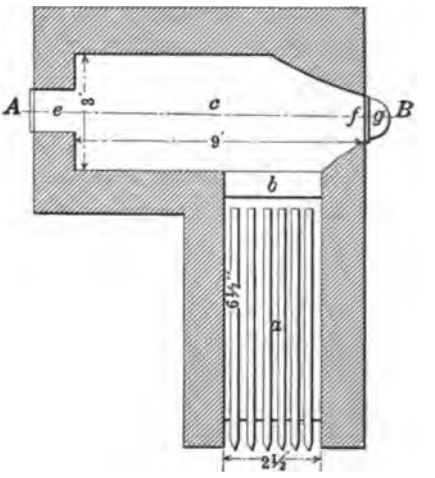
as carried out in the so-called air-furnace, is given here as the American improvement of the Raibl furnace.

The galena ore from the Mississippi Valley is concentrated to a high-grade pure product ranging from 70 to 84 per cent. Pb.

The furnace has a peculiar construction. The drawings (Figs. 40 and 41) show an inclined hearth, *c*, with *e* as charging and working door, and *f* as discharging and cleaning door; beneath is a small kettle, *g*, into which the lead flows as it is set free in the furnace. The chimney is a sheet-iron pipe. Thirteen inches from the front—*i.e.*, the lower or pot end of the furnace —is the firebridge, *b*, with fireplace, *a*, at right angles to the axis of the furnace. The bottom of the



VERTICAL SECTION ON LINE *A-B*



HORIZONTAL SECTION ON LINE *C-D*

FIGS. 40 and 41.—Air furnace, Missouri.

furnace is a cast-iron plate with 6 in. of gray slag or residue melted upon it. Cordwood is used as fuel. The cost of a furnace, including shed, is estimated by Broadhead at $550.

The charge, from 1400 to 1600 lb. of galena of from pea to hazelnut size, is introduced through door *e* and spread evenly over the hearth. It is roasted for from 1 to 1½ hr. at a low temperature, the shorter time being sufficient if the galena contains some oxidize ore. During the roasting the charge is constantly raked and moved from firebridge to flue, and from the cooler part of the furnace to the hotter, in order to heat and roast it as uniformly as possible. When this is accomplished the heat is raised and lead begins to flow. Ashes and breeze are used as stiffening ingredients. The charge is rab-

nated; but even in 1890 it had given way largely to the ore hearth, as this has about three times the capacity, although it recovers a smaller amount of lead. The percentage of lead ore worked in 1880 by different furnaces in the Mississippi Valley, as reported by the "Tenth, Census of the United States," vol. xv, p. 818, was as follows: Air furnace, 10.61; Flintshire furnace, 7.64; Scotch hearth, 62.47; blast furnace, 19.28 per cent.

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bled at intervals. When no more lead appears (after from 7 to 8 hr.) the residue is drawn without any attempt to extract slag-lead in the furnace as at Raibl.

One furnace-man with a helper works in 12 hr. a charge which consumes one and one-half cords of wood. About the yield nothing is known, as the lead-content of ore charged is not determined. Tabulated dimensions and results are given in §63.

The following two analyses by Williams[1] show the composition of residue and poled lead from the Granby works:

Residue: SiO_2 21.396, CaO 4.650, MgO 3.948, Fe_2O_3 3.680, Al_2O_3 0.152, ZnO 7.146 per cent., $PbSO_4$ 2.349, PbS 20.929, PbO 34.914; total, 99.063 per cent. The sample yielded 3.52 per cent. of metallic lead, which makes the total metallic lead 54.82 per cent.

Lead: As 0.01122, Sb 0.00077, Ag 0.00080, Cu 0.05091, Fe 0.01582, Zn 0.00090, Ni 0.00281, Pb (by difference) 99.91777; total, 100 per cent.

61. The English Method.—The characteristics of this method are a large charge, a quick roast (with the result that for every part of $PbSO_4$ formed

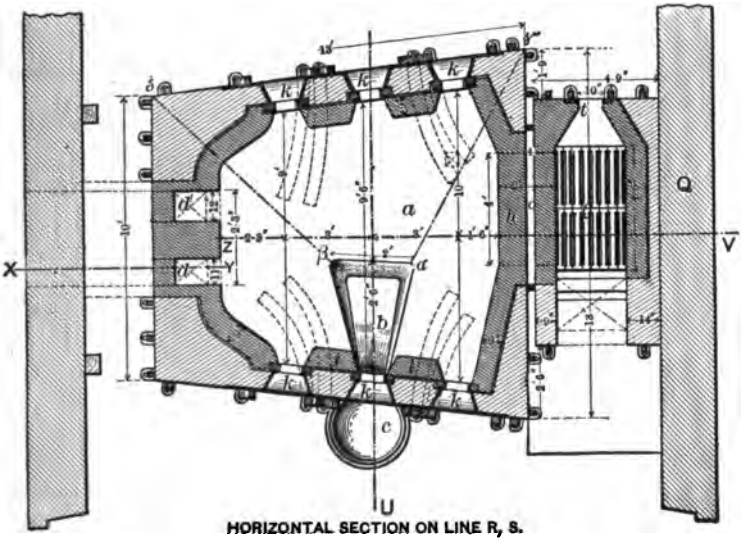

FIG. 42.—Reverberatory furnace, Stiperstones, England.

there shall remain two parts of PbS unchanged), a high temperature throughout, and the aim to extract all the lead in the reverberatory. The hearth inclines toward the middle of one of the sides; the lead collects in the furnace, and is tapped at intervals into an outside kettle.

(*a*) *Lead Smelting at Stiperstones*[2] (*Shropshire*).—The ore is a concentrated

[1] *Tr. A. I. M. E.*, 1876–77, V, 320–324.

[2] Moissenet, *Ann. Min.*, 1862, I, 45; reprint, "Traitement de la galène au four gallois," Dunod, Paris, 1862; also in *Berg. Hüttenm. Z.*, 1863, XXII, 243, 251, 261, 265.

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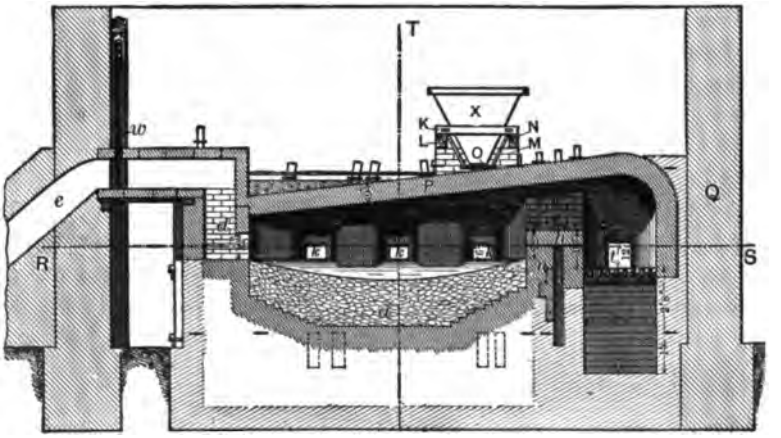
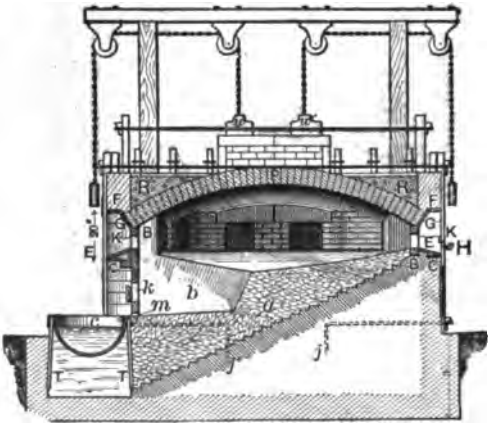
galena with some carbonate and blende, assaying 77.5 per cent. Pb, the gangue being principally carbonate of lime.

The construction of the furnace is given in detail in Figs. 42 to 47. The horizontal section (Fig. 42) shows the usual trapezoidal form of the English reverberatory with its three working doors, *k*, on either side, the well, *b*, at the front, and the fire-opening, *t*, at the back. It is to be noted that the centers of



LONGITUDINAL SECTION ON LINE X, Y, Z, V.



CROSS SECTION ON LINE T, U.

FIGS. 43–44.—Reverberatory furnace, Stiperstones, England.

hearth, *a*, and fireplace, *g*, are not opposite each other, the latter, with fire-bridge, *h*, being 10 in. further back. The object of this is to draw away the flame from the well, *b*; for the same reason the flue, *d'*, near the back, is made 1 in. wider than flue *d*, near the front, while both have the same height. Each has its own damper, *w* and *w'*. The cross-section, Fig. 44, shows the inclination of the hearth from the back to the front (24°), where well, *b*, is placed.

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This has nearly vertical walls and (Fig. 42) extends 3½ ft. into the furnace, where it is 2 ft. wide, narrowing down to a few inches in the front. The lead collected in the well is tapped through tapping-hole, *m*, into the cast-iron kettle or pot, *c*. The longitudinal section (Fig. 43) shows the inclination of the hearth, which is 30° from the bridge and 23° from the flue. The roof, *P*, slopes in a straight line from top to grate, *g*, to flues, *d d'*. The gases from the grate pass over firebridge, *h*, to the hearth and thence through flues *d d'* into flue *e*. In the front elevation (Fig. 45) can be seen the chimney, *f*, which is 2 ft. 6 in. by 2 ft. in the clear and 20 ft. high from the ground. It does not communicate

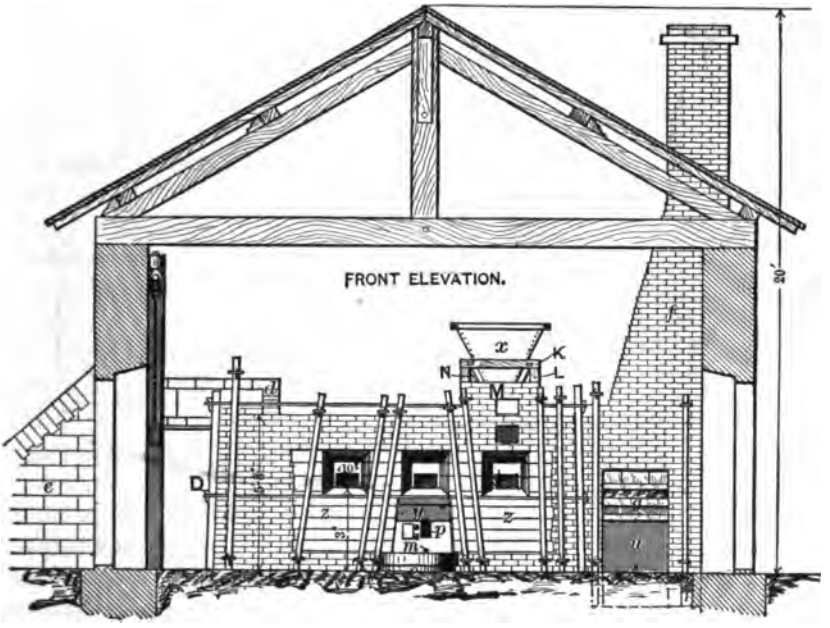

FIG. 45.—Reverberatory furnace, Stiperstones, England.

with the interior of the fireplace. It carries off gases that would enter the smelter building when the fire is fed or stoked or when the doors are open, and also vapor that comes from the ash-pit, *u* (Fig. 43), which is filled with water.

The following details will elucidate the drawings. In Figs. 42, 43, and 45 one end of the furnace is seen to be built against the wall, *Q*, of the building. At the others there is a passage, *D*; 15 ft. of free space at front and back give sufficient room for working the furnace.

The furnace is built of red brick and lined with a full course of firebrick, which is indicated by heavy shading. The foundation is 3 ft. deep; the space in the center between the bent ends (*j j'* in Fig. 44) of the lower tie-rods is filled with sand. The upper part of the foundation is solid and slopes from the back and both ends to the front. The edges of the last course of brick are left sharp, as shown by the zigzag line in Figs. 43 and 44. In determining the slope,

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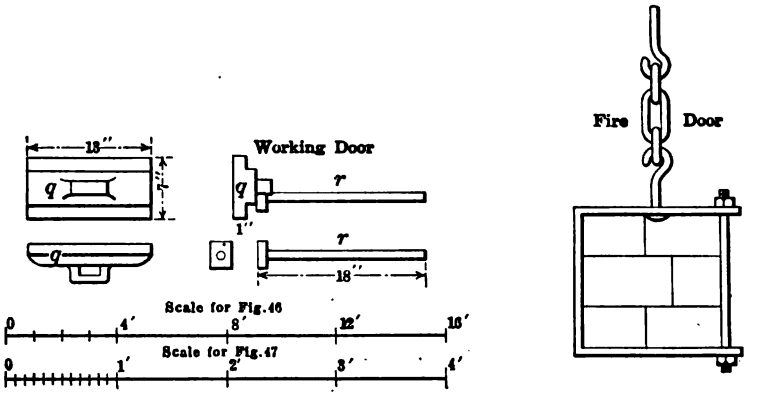
a line 3 ft. 6 in. long is drawn from the taphole, and the points, α and β, are located at a distance of 1 ft. on each side of its inner extremity. To them, lines are then drawn from the corners, γ and δ, starting at an elevation of 2 ft. 8 in. The side walls are raised slightly over the roof, *P*, and have a total height of 5 ft. 8 in., measured from the floor. The space between the two is filled with sand, *R'*.

The ironing of the furnace is clearly shown in the figures. The buckstays are 6 ft. 6 in. long, the tie-rods being slipped over them and tightened with wedges. The upper tie-rods reach over the furnace; the lower ones, *j j'*, are turned down 1 ft. at a distance of 4 ft. 6 in. The castings are not shown with sufficient distinctness in the drawings.



FIGS. 46 and 47.—Working and fire doors, reverberatory furnace, Stiperstones, England.

The taphole plate (Fig. 44 and *y* in Fig. 45) is 5 ft. long and 20 in. high; it is 2 in. thick at the bottom, increasing to 3 in., which it reaches at a height of 6 in. and retains for the rest of the distance. Eight inches above the center of the lower edge is the taphole, *m*, and 4 in. above this begins a narrow vertical opening (4 by 9 in.), with a hinged door, *p*, only 8 in. high, so that when the door is closed a small current of cold air may pass over the molten lead in the well. The taphole plate is overlapped on each side by five cast-iron plates (4 ft. long and ½ in. thick), placed horizontally (*z* and *z'* in Fig. 45). They extend from the working opening down to the taphole.

The back-plate (5 ft. long, 20 in. high, and 1 in. thick) is placed just opposite the taphole plate. Its main object is to protect the brick from the gray slag which is drawn out from the back door. To facilitate this, two hooks, *H* (Fig. 44), carrying an iron rod (3 ft. 6 in. long and 1 in. in diameter) as support for the working-tools, are placed inside of neighboring buckstays.

The bridge-plate, *n* (Fig. 43), counteracting the longitudinal thrust of the hearth, gives strength to the 4-in. air-space, *o*, and prevents any leakage of lead. It is 6 ft. long and 20 in. high. From the upper edge, which is on a level with the grate bars, it is 3 in. thick for a distance of 6 in., then suddenly increases to double that thickness for another 6 in., and for the last 8 in. returns to its original thickness.

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The working openings, *k*, have a number of castings. The door-frames (*B*, Fig. 44), inclosing each an opening 10 by 6 in., are 4 in. square in cross-section. They are protected on the sides by firebrick, and on the bottom by the hearth. On the level of the grate-bars, or 2 ft. 8 in. above the floor, are two horizontal plates, *C*, 6 in. wide and ½ in. thick, that extend below the six working openings. On them rest two inclined (3 : 8) plates, *E*, 8 in. wide and ½ in. thick, which abut against the bottom of the door-frames. Two plates, *F*, of the same size as *C*, form the upper part of the linings. The skew-backs, *G* (9 in. wide and ½ in. thick), resting on these upper plates, *F*, and the door-frames, *B*, support the roof. The sides of the working openings are lined with ½-in. jamb-plates, as shown in Fig. 42. The cast-iron working doors, *q*, with handle, *r* (Fig. 46), are placed against the door-frames, *B*.

The cast-iron kettle, *c*, in front of the taphole is 2 ft. 6 in. in diameter and 20 in. deep. It is imbedded in clay, which is inclosed on three sides by masonry and on the fourth by the slag-bottom. Its rim, 8 in. above the floor, is surrounded by an iron hoop, 4 in. wide and ½ in. thick. Wedges, *T*, are driven between it and the kettle to prevent any lead that may penetrate through the joints and collect under the kettle, from raising it.

The fire-opening, *t*, 10 by 12 in., has an iron casing (Fig. 42). It is closed by a swinging fire-door (Fig. 47), consisting of firebrick held together by a wrought-iron frame. The grate consists of two sets of cast-iron grate-bars of eight each, supported at the ends by cast-iron crosspieces.

The hopper (*x*, Figs. 43 and 45) is a truncated sheet-iron pyramid 36 and 6 in. square at the ends and 3 ft. high. It is suspended in a wooden frame, *K* (3 in. square), which rests on two iron cross-beams, *L*, 4 by 6 in. These are supported by brickwork, 14 in. high, forming the continuation of the sidewalls of the furnace. Four iron rods, *N* (Fig. 43), fastened in the wooden frame, *K*, have at their lower ends a sheet-iron frame, *O*, on which is placed a movable slide, which closes the discharge of the hopper. The 6-in. opening in the roof of the furnace, over which the mouth of the hopper is placed, is indicated on the hearth in Fig. 42. It is 1 ft. 9 in. distant from the firebridge, and 2 ft. 6 in. from the working opening nearest the bridge at the back of the furnace.

When the furnace is finished, the working bottom, made of coarsely broken gray slag and sand, is put in. Part of the material is spread out on the brick bottom, which has previously been made red-hot. It is heated till it becomes pasty, and is then patted with paddle and rake. A second part is made to adhere to the first, and so on until the entire bottom has been built up with successive layers of gray slag and sand, and has attained the desired form and thickness. The upper edge of the well is 10 in. below the working opening. The thickness of the working bottom increases toward the taphole. It is 4 in. at the back doors, 12 at the bridge, 14 at the flues, 30 at the front, and 8 at the back of the well.

The furnace, being built solidly, lasts a long time; the roof of the fireplace requires repairing every 2 years, that of the hearth every 5 years. The hearth is repaired after every charge if necessary.

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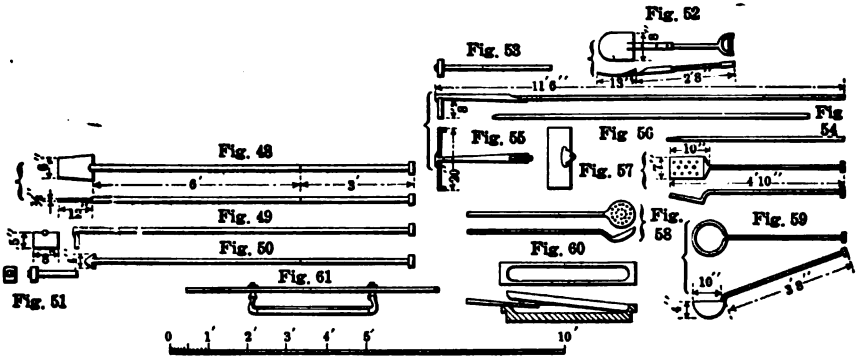
This furnace has the representative form of the English reverberatory. Slight variations exist as regards detail. For instance, the foundation, instead of being built up solid at the back, sometimes has an arched vault[1] at *j* (Fig. 44), extending longitudinally, communicates with the air-space in the firebridge, and is accessible at the flue end of the furnace. Another variation is building the entire hearth on cast-iron plates[2] resting on rails supported by a brick pillar. Sometimes the general form of the hearth differs by having a gentle slope toward the taphole. This makes the sides of the well less steep than those of the drawings.

The tools used in the furnace are shown in Figs. 48 to 61. They are of wrought-iron, with the exception of the mold (Fig. 60) and the handles of shovel (Fig. 52) and ash-pit hoe (Fig. 55).

Figs. 48 to 51: Paddle, rake, old paddle used as chisel, and hammer to remove adhering slag are the tools for working on the hearth. Figs. 52 to 55: Coal-shovel, hammer to break coal, fire-poker, and ash-pit hoe are required for firing. Figs. 56 to 61: Tapping-bar, rectangular skimmer, circular skimmer, ladle, mold, and lead-carrier are used in handling the lead.



FIGS. 48 to 61.—Tools, reverberatory furnace, Stiperstones, England.

Method of Working.—To simplify the description the doors may be designated by numbers 1, 2, 3 in front, and 4, 5, 6 at the back, starting both times from the bridge.

The charge, 2350 lb., is dropped from the hopper through the roof into the furnace, still red-hot from a previous charge, the dampers having been closed. The lead is left in the well and covered with lime. The ore is spread with rakes over the hearth through doors 1, 4, and 5. It decrepitates and gives off vapor of water. Then the dampers are slightly raised and the fire is gradually increased for 1½ hr. During the first 2 hr. (first firing) the ore is turned over four times with paddles through doors 1 and 4, the other doors being kept closed. The paddling requires only a few minutes. When the first firing is nearly accomplished, feeding of fuel is stopped, and doors 1, 2, 4, and 5, fire-door and dampers are thrown open, and the charge cooled (first cooling).

[1] Percy, "Lead," pp. 222–229.

[2] Phillips-Bauerman, *op. cit.*, 1891, p. 640.

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This lasts for ½ hr. The grate is then freed from clinkers, and the charge, nearly 6 in. deep next to the bridge, which has fritted, is broken up and turned over. Any ore that had fallen into the well is raked up on the furnace-bed. The doors are now closed, the dampers lowered, the grate is well filled with coal, and the second firing begins. This lasts from 55 to 60 min., and is followed by a second cooling. Some lead now flows into the well. Near the bridge and toward the center of the furnace, parts of the charge have begun to fuse, whereas at the flue the ore is only sintered. The charge is worked as in the first cooling, door 6 now being also used. After 10 min. the lead which has accumulated is tapped from the well into the kettle. The taphole is closed from the inside by inserting a clay plug fixed to a wooden handle through the little taphole door and pressing it in until the clay oozes out in front. Before tapping, fine coal and wood-shavings are put into the kettle to pole the lead. It is then stirred vigorously for 6 or 7 min. with the rectangular skimmer to bring all the impurities to the surface. These are removed, first with the shovel and then with the rectangular skimmer, and thrown back into the furnace, through doors 1 and 3, on both sides of the well. The charge, now freed from part of its lead, is turned over with paddles through doors 1 and 4. This ends the second cooling, 4 hr. after charging. The doors are now closed, coal is put on the grate, and the dampers are lowered to begin the third firing. Meanwhile the lead in the outside kettle is skimmed and ladled into molds holding 120 lb. This takes about 20 min. During the third firing, which lasts 2 hr., the ore is turned over several times, care being taken to open the doors as little as possible. The furnace shows a full-yellow heat when the third cooling begins. The charge is now worked with paddles for 15 or 20 min., and parts of it that have collected in the well are stiffened by the addition of lime and are raked on the hearth. The residue on the hearth is collected near the bridge and fine coal worked into it. The doors are again closed, the fire is urged for a quarter of an hour, and the residue turned over with the paddle through doors 1 and 4, and finally drawn out through door 5. Any repairing of the hearth that may be necessary takes place now, and the furnace is ready again for a new charge, 7 hr. from the time when the previous one was first introduced.

Two men work as partners in 12-hr. shifts. Tabulated results are given in §63.

(*b*) *Smelting Near Holywell, North Wales.*—This differs from that at Stiperstones. According to Percy[1] it is as follows:

After dropping the charge of 2350 lb. on the hearth and spreading it over the upper part of the bed, the doors of the furnace and that of the fire-box are left open for 1½ hr. to let the air have free access during the first roasting stage, while the damper is raised just enough for the gases to escape. Working doors 3 and 6 and the fire-door are now closed and the fire is urged. Lead soon appears. During this heating, which lasts 2 hr., doors 2 and 5 are closed, but 1 and 4 are kept open. Through these the charge is rabbled at

[1] "Lead," p. 232.

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intervals. Toward the end of this first reaction period lead begins to flow. Now doors 1 and 4 are closed, the damper is thrown open, the fire urged for 40 min., and the charge melted down. The furnace is then cooled for half an hour by throwing open all the doors. What charge remains on the hearth is rabbled, doors 3 and 6 are closed, slaked lime is thrown through door 2 on the charge, which has collected in the well, and worked into it through the taphole door. The stiffened residue is collected ("set up") near the bridge, as well as other parts that have been detached from the hearth. Doors 2 and 5 and the fire-door are now closed, the damper is lowered, and the temperature raised gradually for half an hour, whereupon the damper is entirely thrown open, doors 1 and 4 are closed, and the fire is urged to melt down the residue, which takes 20 minutes. The fire-door and the working doors 1, 2, 4, and 5 are then thrown open, lime is added through door 2, and worked into the slag to thicken it. The lead is tapped, and the stiff gray slag raked out on the floor through door 5. The lead is poled as at Stiperstones, and the hearth repaired, if necessary. The entire time required to work the charge is 6 hr.

Two furnace-men and one helper work two charges in a 12-hr. shift. Tabulated results are given in §63.

The changes that take place in the ore during the process have been examined by Percy,[1] and are given here in their lead contents only.

TABLE 25.—ANALYSES OF CHANGES OF GALENA IN ROASTING AND OF GRAY SLAG

Lead compounds	Hours after charging				Gray slag
	1½	3½	4½	4¾	
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
PbS	63.82	53.32	24.76	4.35	0.90
PbO	27.25	31.49	43.12	47.50	48.87
$PbSO_4$	3.82	4.78	6.94	14.02	9.85
Total Pb	83.16	78.66	66.22	47.86	52.88

(*c*) *Smelting at Desloge, Flat River, Mo.*[2]—This plant has one English and three Silesian reverberatory furnaces for ores, and one blast furnace for reverberatory slags which are too rich to be discarded. In 1914[3] only two reverberatories were in operation, and the gray slag was sold to other smelteries.

The Silesian furnace, which was at first constructed on the model of the one shown in Figs. 62–67, was not successful, probably on account of the lack of the skilled labor required. It was therefore rebuilt with the sump in the middle of one side, following the example of the English reverberatory; but the rectangular Silesian form was retained.

[1] *Op. cit.*, p. 235.

[2] Ingalls, *Eng. Min. J.*, 1905, LXXX, 1111.
Pulsifer, *Min. Eng. World*, 1914, XL, 1152.

[3] *Min. Res. U. S.*, 1914, I, 102.

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The hearths of the two remodeled furnaces are 16 ft. long and 11 ft. wide, that of the third furnace is 16 by 13 ft. The width of 13 ft. is too great for satisfactory work; it is held that a width ought not to exceed 10 ft. Of the four working doors on a side, the pair near the flue is kept closed on account of insufficient temperature in that part of the furnace, *i.e.*, the furnace is too long for the English method of working.

The hearth 16 by 11 ft. is fired from a grate 8 by 3 ft.; the ratio of hearth to grate area is therefore 7.3 : 1. The grate lies 6 in. below the fire-door; the bars are 36 in. long, 1 in. thick at top, 4 in. deep at middle, and 2 in. deep at ends; the open spaces, including rib in middle, are 32 in. long and 5/8 in. wide. There are two flues, 16 by 18 in., each with a damper. The sheet-steel guyed chimney, 6 ft. from the furnace, is 42 ft. high and 42 in. in diameter, and has a brick pedestal, 64 by 64 in. and 13 ft. high, making the total height 55 ft. The working doors, 36 in. above the floor, have cast-iron frames, 7 by 11 in. on the inside and 15 by 28 in. on the outside, and are held in place by buckstays. The sump holds a cast-iron kettle of 1/4-in. metal, which is set partly below the floor-line so that the distance to the working door is 31 in. and to the tapping door 7 in.; the latter forms an opening 4 by 9 in. in the 1.5-in. cast-iron plate. The taphole, 3.5 in. in diameter, is on a line with the rim of the kettle. The furnace costs $2000 excluding the chimney. The working hearth, 8 in. thick, is of broken gray slag beaten down firmly on a layer of brick which rests on a filling of rammed clay. Firing a furnace, after repairs, for from 8 to 12 hr. brings it to the proper temperature; Illinois coal serves as fuel. The method of working is that of the English reverberatory. The ore is a galena concentrate with about 70 per cent. Pb. A furnace is manned by a smelter and a helper who get their ore, flux, and fuel, and wheel away the gray slag and ashes. With three 8-hr. shifts, each working a charge of 3500 lb. wet (3325 lb. dry) concentrate, the schedule of operations is the one given in Table 26.

TABLE 26.—SCHEDULE OF REVERBERATORY OPERATIONS AT DESLOGE

Drop charge	7 a.m.	Begin tapping	9 p.m.
Begin firing down[1]	12 noon	Rake out slag	10
Begin tapping	1 p.m.	Begin tapping	10.30
Rake out slag	2 p.m.	Drop charge	11.00
Begin tapping	2.30	Begin firing down	4 a.m.
Drop charge	3.00	Begin tapping	5
Begin firing down	8.00	Rake out slag	6
.....	.....	Begin tapping	6.30

At the beginning of the reaction period, that is 5 hr. after dropping the charge, from 85 to 100 lb. crushed fluorspar is thrown into the furnace and worked into the charge, the doors are closed, and the fire is urged. The first tap yields about 1200 lb. lead. Chips of wood and bark are placed on the lead, which is poled and then ladled into pigs weighing about 80 lb.; the

[1] "Firing Down" at the beginning of the reaction period means raking out the entire bed of coal and then building a fresh fire with cordwood and lump coal.

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dross and skimmings are returned to the furnace. At the end of the shift, the gray slag is raked out into a barrow and wheeled outside the building. It is sold with a payment of 95 per cent. of the Pb-content if it assays over 40 per cent. Pb, and 90 per cent. if under. The coal consumption is 0.8–0.9 ton per ton of ore. Treatment of 1012 tons wet (=962 tons dry) concentrate with 70 per cent. Pb or 673.4 tons lead, gave 523.3 tons or 54.4 per cent. pig lead; 262.5 tons or 27 per cent. gray slag, assaying 38 per cent. Pb, which is equal to 99.66 tons lead. Assuming 90 per cent. extraction by smelting in the blast-furnace, gives 89.7 tons lead. The total extraction is therefore 523.3+89.7= 613.0 tons lead, or 613.0×100=673.4=91 per cent. Tabulated results are given in §63.

The cost of smelting in 1905 with five furnaces, each treating three charges in 24 hr. is given in Table 27.

TABLE 27.—COST OF REVERBERATORY SMELTING AT DESLOGE

Labor and materials	
1 foreman at $3.00	$ 3.00
5 crews at $9.90	49.50
21 tons coal, unloading at 6 cts.	1.26
14 tons lead, loading at 15 cts.	2.10
7 tons gray slag, loading at 15 cts.	1.05
Total labor	56.91
Coal, 21 tons at $2.00	42.00
Flux and supplies	13.00
Blacksmithing and repairs	10.00
Grand total	$121.91

On the basis of 6.25 tons wet concentrate per charge, this cost would be $4.65 per ton. The actual cost for 7 months in 1900 was per ton: Labor, $1.98; coal, $1.86; flux and supplies, $0.51; miscellaneous, $0.017; total, 4.757. Figuring the cost of smelting gray slag at $8.00 per ton, and the production of gray slag at $0.25 per ton of wet concentrate, the total cost per ton wet charge is about $6.75, or per ton dry charge, $7.00.

Smelting in the reverberatory furnace was replaced in 1905 by blast roasting in Huntington-Heberlein pots (§93) and smelting in the blast furnace.

Birnbaum[1] draws a valuable comparison between the working of the two processes at Tarnowitz, Silesia.

62. The Silesian Method.—The characteristics of this method are a large charge, slow roasting, and a low temperature. It is not aimed to extract all the lead in the reverberatory, as this is supplemented by the blast furnace. The hearth is inclined toward the flue, beneath which the lead is collected and tapped at intervals into an outside kettle.

[1] *Zt. Berg. Hütten. Sal. Wes. i. Pr.*, 1905, LIII, 219; *Eng. Min. J.*, 1905, LXXX, 535, 680; *Min. Ind.*, 1905, XIV, 405.

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Lead-Smelting at Tarnowitz,[1] *Prussia.*—The ore is a mixture of sulphide and oxide lead minerals. Its composition, in 1870, in round numbers is given in Table 28.

TABLE 28.—COMPOSITIONS OF ORE MIXTURES AT TARNOWITZ

	Per cent.	Per cent.
PbS	61	40
$PbSO_4$	11	9
$PbCO_3$	24	45
(Ca, Mg, Fe, Zn) CO_3	3	5
SiO_2	1	1 and less.

The furnace charges contained from 70 to 74 per cent. Pb and from 21 to 22 oz. Ag per ton. At present the ore runs lower, the percentage of blende having increased considerably.

The construction of the latest furnaces is given in Figs. 62 to 67. After the detailed description of the reverberatory at Stiperstones attention need be called to some of the principal features only. The horizontal section (Fig. 62) shows the rectangular form of the furnace with four working openings on either side, the well being below the door nearest the flue end, the coolest part of the furnace. The grate bars are placed parallel with the axis of the furnace. The fire is stoked at the end and fed from both sides. Four branch flues lead the gases into one main flue. The bridge-wall has an air-passage. The entire furnace is incased in iron plates and bound by buckstays (iron rails) and tie-rods. In the longitudinal section, Fig. 63, the roof is seen to form a horizontal line from the fireplace to the beginning of the hearth, and to slope thence in a straight line to the flues. The hearth, in Figs. 63 to 65, is built up differently from that of any other furnace. First a layer of sand is tamped down between the furnace walls so as to have the form of a trough inclining from bridge to flue. At the well the brick bottom, of red-brick set dry, replaces some of the sand. It forms a good support for the brasque bottom, which is followed by the slag bottom, consisting of tap-cinder melted down in the furnace. This covers the brasque bottom, wherever it would otherwise come in contact with working tools, leaving it exposed only at the well (Fig. 64).

During the run some residue adheres to this slag bottom and forms the smooth working bottom of the furnace. The hearth is seen in Fig. 65 to be trough-shaped from the bridge to the second working door; from there the

[1] Teichmann, *Zt. Berg. Hütt. Sal. Wes. i. Pr.*, 1866, XIV, 225.
Wedding, *op. cit.*, 1871, XIX, 157.
Dobers-Dziegiecki, *op. cit.*, 1884, XXXII, 94.
Dobers-Althans, *op. cit.*, 1886, XXXIV, 292.
Saeger, *op. cit.*, 1893, XLI, 267.

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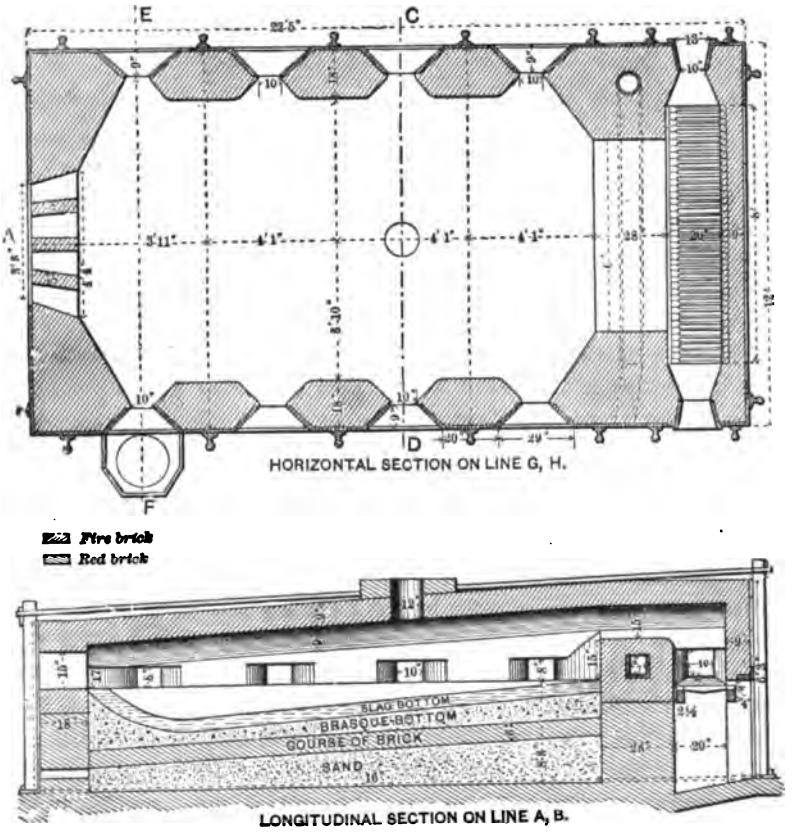
front part (Fig. 64) slopes down to the level of the rim of the kettle, the lowest part of the well.

The latest improvements of the Tarnowitz furnace, shown in Figs. 66 and 67, have been devised to make the work less dangerous and unhealthy. The gray slag, when drawn from the furnace, is not now discharged on to the floor or into a water-box, exposing the laborers to heat and fume or to explosions, but into a slag-pot, *d*, placed in a niche, *a*, where the slag may cool and the fumes pass back into the furnace through the opening. Further, the fumes issuing



FIGS. 62 and 63.—Reverberatory furnace, Tarnowitz, Silesia.

from the basin after the lead has been tapped into it do not pass into the building, but are drawn off by a telescope stack, *h*. The niche, *a*, 30 in. wide, 26¼ in. deep, and 26¼ in. high, forms an opening beneath the drawing door, *b*, (to receive the slag-pot, *d*, which is brought on a separable car), the back and sides of which are inclosed by brickwork, the roof being formed by a heavy cast-iron plate which supports the brickwork and the hearth. The plate has an opening, *c*, 7½ in. square (beginning 15 in. from the outside wall), through which the gray slag is raked into the pot. It is closed with an iron plate when not in use. When the slag is being drawn, the front of the niche is closed by a movable

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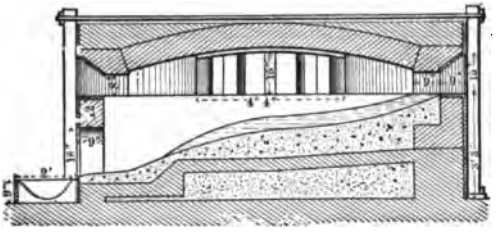
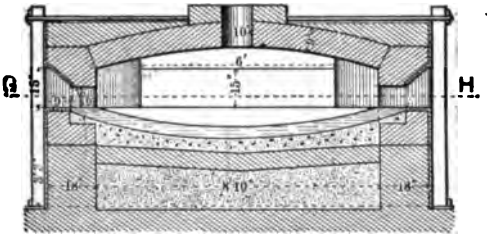
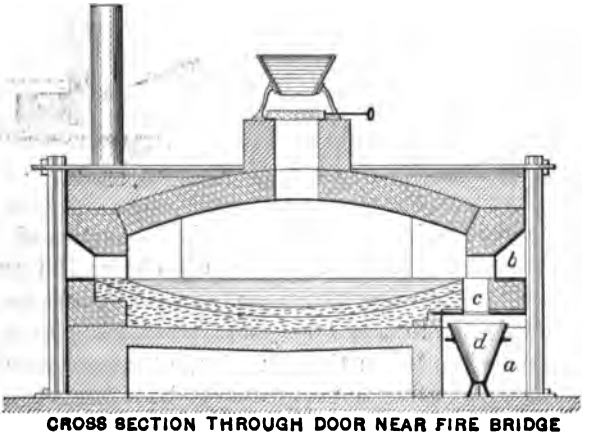
sheet-iron door, having a peephole; all the fumes from the slag pass through the drawing opening, *c*, back into the furnace. When it has cooled sufficiently, the slag-pot is covered with a piece of sheet iron and run out on the car. The telescope stack, *k*, which is balanced by counterweights, is lowered to the position, *k*, as soon as the lead is to be tapped, when the fumes will pass off through



VERTICAL SECTION ON LINE E, F.



VERTICAL SECTION ON LINE C, D.



CROSS SECTION THROUGH DOOR NEAR FIRE BRIDGE

FIGS. 64, 65 and 66.—Reverberatory furnace, Tarnowitz, Silesia.

the stationary sheet-iron pipe, *g*. The beneficial effect of these improvements is shown by the fact that while in the year 1887–88, 28.5 per cent. of the men became leaded, in 1891–92 only 0.8 per cent. were thus affected.

The tools required to work a furnace are four paddles, four large and two small rabbles, five shovels (two for lime, two for coal, and one for slag), three steel bars (two large ones and a small one), a tapping-bar, a skimmer, a ladle,

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a sample-ladle, two slice-bars, a sledge, two hammers, and four hooks for handling the lead.

Method of Working.—The furnace, if new, is brought to a dark red heat; the damper is then closed and the charge, crushed to pass a 5-mesh sieve, dropped from the hopper through the opening in the roof and spread out evenly over the hearth by means of rabbles to a thickness of from 3 to 4 in. The fire is fed with cinders, and the temperature never allowed to exceed dark redness, say 500° or 600° C. The galena decrepitates, the temperature rises,

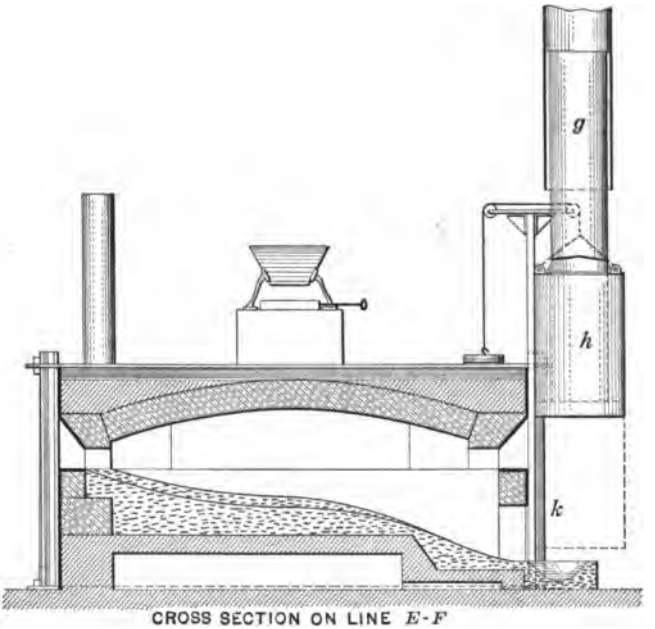

FIG. 67.—Reverberatory furnace, Tarnowitz, Silesia.

the ore becomes a dark-red, and after from three-quarters of an hour to an hour the roasting begins. During this time the charge is turned over once with the paddle. The working doors and fire-doors are opened, and the damper raised sufficiently to allow SO_2 and other gases to pass off. The ore roasts on the surface. When the fumes begin to lessen, samples are taken to see if a white crust of oxide and sulphate has been formed. This indicates that it is time to renew the surface, which is generally done by paddling or rabbling every 20 or 25 min., *i.e.*, about eight or nine times during the 3 or 4 hr. required for roasting. Care is taken to prevent the ore from clotting.

Up to 1885 the roasting usually took about 3 or 3½ hr. but the gradual increase of blende in the ore brought the required time up to 4 hr. In 1886 it became necessary to make a change in working the charge. This could be done in two ways. Either the normal time could be prolonged or part of the ore roasted separately. The latter method was chosen, and now one-quarter of the charge, fine concentrates especially rich in blende, is roasted separately

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in a long-hearth reverberatory furnace and then added to the ore charge. Toward the end of the roasting period from 330 to 660 lb. of carbonate ore or flue-dust, with 45 per cent. Pb, are added to increase the amount of oxides in the charge.

The grate is now cleaned, well filled with good coal, and the damper opened to raise the temperature. The reactions set in, the roasted ore gradually softens, white fumes are given off, and lead begins to flow. Lime is added to prevent the lead from carrying down to the well particles of ore floating on top of it and to prevent liquefying of the charge. The lime is well worked in and the charge then turned over. The furnace becomes filled with fumes of lead, the damper is raised to remove these, and fusing of the charge is prevented by regulating the fire and the damper and by adding a few shovelfuls of lime at a time. At from an hour to an hour and a quarter after the reaction has set in (3 hr. with a new furnace) the well is full of lead. This is tapped into the outside kettle, after the telescope stack has been lowered. The dross floating on the surface is skimmed off and put back into the furnace, and the impurities held in suspension by the lead are removed by stirring in slack coal (poling). This second dross is put aside and the lead ladled into molds. During the reaction stage the working doors have to be open on account of the necessary rabbling and paddling; thus air enters, cools the furnace, oxidizes the charge, and the flow of lead begins to cease.

At this stage drosses obtained from melting down lead bullion in the desilverizing plant are added. They consist principally of a mixture of PbS and Pb, assaying 75 per cent. Pb and 9 oz. Ag per ton. At the same time 1,100 lb. of oxides low in Ag, containing from 75 to 80 per cent. PbO and from 19 to 20 per cent. ZnO are charged. They result from refining lead, desilverized by the Parkes process, by means of steam (§252).

The doors are then closed and fuel is piled up high on the grate to insure a smoky flame. This reduces $PbSO_4$ to PbS, and soon the flow of lead begins again. The doors are opened, the charge is worked, and its liquefying prevented as before. From 2½ to 3¼ hr. after the first reaction sets in, the well is again filled with lead. This is tapped, and lead and dross are treated as before. In this way all together five reactions are caused, the amount of lead obtained becoming gradually smaller and the charge more dry. Before the last reaction takes place the mixture of dross and fine coal, obtained from poling the different leads in the kettle, is thrown into the furnace. It is worked into the charge, which is then covered with fine coal. The temperature is raised and the last yield of lead obtained. The damper is now opened fully and the residue drawn from the back of the furnace through opening *c* (Fig. 66) into the slag-pot. Formerly it was drawn into a box filled with water, so as to prevent fumes of lead from passing into the building, and care had to be taken to avoid explosions. The hearth is repaired with a mixture of crushed residue and slaked lime, which is beaten down with the paddle and rammed into holes with a bent bar. When this is finished, a little lime is spread over the hearth and a new charge dropped from the hopper. The whole reaction period lasts 7 hr.

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If the hearth should become so incrusted as to raise the charge too high to be protected by the firebridge from the direct action of the flame, the temperature must be raised so as to soften the accumulated residue that it may be removed. A hearth lasts about 3 months.

The results from 1 year's (1865) work in a smaller furnace, 11 ft. 9 in. by 10 ft. 10 in., with six doors, and with ore low in blende and without the different additions, were: 10,000 lb. of ore (assaying 72.97 per cent. Pb and 21.76 oz. Ag) gave 6384 lb. lead bullion (assaying 32.95 oz. Ag); 1592 lb. residue (38.80 per cent. Pb and 3.93 oz. Ag); 275 lb. flue-dust (50.00 per cent. Pb and 0.02 oz. Ag); showing that in the reverberatory 87.49 per cent. of the lead and 99.9156 per cent. of the silver was recovered in the form of lead bullion. Of the 12.51 per cent. of lead, forming the difference, 9.31 per cent. was recovered in the blast furnace, so that the actual total loss amounted only to 3.2 per cent.

The consumption of fuel was 4,600 lb., or 46 per cent. of the ore charged, and of lime 100 lb., or 1 per cent. Tabulated results are given in §63.

The following analysis of residue was made by Pietsch in 1865: PbO 24.375, $PbSO_4$ 13.269, $PbSiO_3$ 12.373, ZnO 22.857, FeO 8.957, FeS 1.823, CaO 11.190, C 4.821, Al_2O_3 and MnO traces, Ag 0.015 (=4.360 oz. per ton)—total 99.680 per cent.

The flue-dust, in 1881–82, forming 2.91 per cent. of the ore charged, had according to Dobers and Dziegiecki the composition given in Table 29.

TABLE 29.—COMPOSITION FLUE-DUST AT TARNOWITZ

	I	II	III
PbO	66.44	66.53	65.79
ZnO	3.77	4.55	4.05
Fe_2O_3	0.50	0.30	0.35
SO_3	27.48	27.11	27.79
Insol	2.14	1.20	2.30
Totals	100.33	99.69	100.28

63. Comparison of Reverberatory Methods.—To facilitate comparison, the main data of the furnaces discussed have been brought together in Table 30.

In comparing the amounts of ore treated in 12 hr. at the different smelting works, the order in which they are placed in the table shows a steady increase from Raibl to Tarnowitz. The figure for Tarnowitz, 8250 lb., requires the explanation that the charge contains a considerable proportion of oxide ore, which shortens the time required for roasting. If the ore were pure galena, twice the time, or 8 hr., should be allowed. This would make the amount for 12 hr. 6187 lb. The large amount of ore treated is due to the size of the furnace. As to the amount of labor required per ton of ore, the table shows that Tarnowitz uses less and Raibl more than any of the other smelting works. With fuel the same is the case. As the wear and tear of a furnace depend on

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TABLE 30.—EXAMPLES OF SMELTING IN REVERBERATORY FURNACES

Items	Carinthian method					English method			Silesian method	
									Tarnowitz, Silesia	
	Raibl, Carinthia	Engis, Belgium	Granby, Missouri	South-west Missouri	South-east Missouri	Holywell, Flintshire	Stiper-stones, Shropshire	Desloge, Missouri	Old furnace	New furnace
The furnace				*						
Hearth, length	11′	6′ 6¾″	9′	9	10′	12′	9′ 9″	16′	11′ 9″	16′
Hearth, width at bridge	5′	4′ 3⅛″	3′ 2″	3′	2′ 6″	3′ 6″	4′	.....	.....	6′
Hearth, width at middle	5′	4′ 4¼″	3′ 2″	3′	4′	9′ 6″	9′ 6″	11′	10′ 10″	8′ 10″
Hearth, width at flue	1′	2′ 3½″	3′ 2″	3′	2′	4′	3′ 3″	.....		4′ 4″
Hearth, thickness of bed	6″	1′ 5¾″–2′ 8¾″	6″	6″	.....	2″–10″–26″	4″–18″–30″	8″		4″
Hearth, life of bed, weeks	4–5	6	.....	.....	.....	.....	.....	.....		13
Hearth, inclination to flue or side	21″ F.	17″ F.	18″ F.	—F.	—F.	29″ S.	24″ S.	.....		27½″ F.
Grate, length	7′	3′ 7¼″	5′ 6″	6′ 6″	3′	4′ 6″	4′ 6″	8′		8′
Grate, width	1′ 3⅜″	1′ 6″	2′ 6″	2′ 6″	2′	2′ 6″	2′ 6″	1′ 3″		1′ 8″
Grate, depth below top of bridge	1′ 2⅜″	3′ 7¼″	.....	6″	.....	2′ 9″	1′ 4″	.....		3′
Bridge, width	5⅜″	1′ 3¾″	8″–30″	.....	21″	24″	24″	.....		28″
Bridge, length	3′ 7¼″	3′ 11¼″	30″	30″	24″	42″	48″	.....		72″
Bridge, height above hearth	4–5″	none	.....	6″	.....	12″	12″	.....		15″
Roof, height above bridge	9″	8⅝″	14″	.....	6″–8″	9″	19″[1]	.....		12″
Roof, height above hearth at flue	12″	2′ 3⅛″	14″	.....	6″–8″	16″	20″	.....		17″
Flue (leading to chimney), size	1′ × 1′ 3¾″	8¾″ × 2′ 3½″	8″ × 12″	.....	12″ diam.	12″ × 10″ / 10″ × 10″	12″ × 12″ / 11″ × 12″	2(16″ × 18″)		4(15″ × 20″)
Chimney, inside size	1′8″ × 1′8″	1′ 3″ × 1′	.....	.....	12″ diam.	3′ × 3′ 6″	.....	42″ diam.		.....
Chimney, height	20′	.....	.....	20′	15′	.....	.....	55′		.....
Grate area, ratio to hearth area	1:6	1:5.25	1:2.0	1:1.66	1:5.4	1:8	1:6.6	1:7.3		1:10
Grate area, ratio to section of chimney	1:0.14	1:0.47	.....	.....	1:0.13	1:0.93	.....	1:0.40		.....

Similar to new furnace.

* Kemp, *School of Mines Quarterly*, 1887–88, IX, p. 212.

[1] Height of roof above grate, 38″.

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TABLE 30.—EXAMPLES OF SMELTING IN REVERBERATORY FURNACES—*Continued*

Items	Carinthian method					English method			Silesian Method	
									Tarnowitz, Silesia	
	Raibl, Carinthia	Engis, Belgium	Granby, Missouri	South-west Missouri	South-east Missouri	Holywell, Flintshire	Stiper-stones, Shropshire	Desloge, Missouri	Old furnace	New furnace
Work done										
Charge, weight in pounds	370-430	1,375	1,500	1,500	.....	2,350	2,350	3,325	4,400	8,250
Charge, assay, per cent. lead	70	76	.....	.....	.....	75–80	77.5	70	70-74	.....
Charge, thickness on hearth	1⅛″–1½″	about 3″	.....	.....	.....	1⅝″?	3⅞″–4⅛″	.....	3″–4″	3″–4″
Operation, length in hours	10–12	12	12	9–11	.....	6	7	8	12	12
Men, number in 24 hr	1.5	4	2	2	.....	6	4	6	4	6
Gray slag, amount in pounds	95	275	.....	.....	.....	288	.....	906	600	2,475
Gray slag, per cent. of charge	24	12	.....	.....	.....	12	.....	27	15	30
Gray slag, assay, per cent. lead	4	17–20	55	.....	.....	55	.....	38	38.8	56
Of lead charged, recovered per cent., in bars	93	92.5	.....	.....	.....	81	88	54.4	87.5	50
Recovered from gray slag and flue-dust	.....	2.3	.....	.....	.....	8	.....	10	9.3	.....
Total loss in lead	7	5.2	.....	.....	.....	11	12	9[1]	3.2[1]	.....
Outlay for labor, fuel and material per ton of ore of 2,000 lb. avoirdupois										
Labor, hours	90	61	36 aver.	20	.....	15	12	8.4	11	9
Fuel, charcoal, pounds	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Fuel, bituminous coal, tons	.....	1.10	.....	.....	.....	0.57–0.76	0.56	0.8–0.9	0.46	.....
Fuel, wood, cords	1.60	.....	1.10 aver.	0.83	.....	.....	.....	.....	.....	.....
Material, lime, pounds	.....	.....	.....	.....	.....	.....	.....	26	20	.....

[1] Includes smelting residue in the blast furnace.

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the height of the temperature, the Carinthian and Silesian furnaces will outlast the English, other things being equal. If, finally, the amount of lead recovered is considered, it is clear that slow roasting yields more than quick roasting (Carinthian and Silesian *vs.* English), and that recovering in the reverberatory only that amount of lead obtainable at a low temperature and not melting the charge, gives a higher percentage of lead (Tarnowitz and Engis *vs.* Raibl and English methods). The inferences to be drawn are clear. They are stated by Cahen,[1] and Grüner[2] has formulated them at length. He says (Percy's translation):[3] "Rich, pure, nonquartzose ores ought always to be treated by this latter (the roasting and reaction) method. The operation ought to take place in large reverberatory furnaces, with easy access of air, provided with a single fireplace and a receiving basin, internal or external, placed in the least heated region of the furnace. The operation ought always to be conducted slowly, and to consist of two phases very distinct, roasting and reaction. For roasting, the layer of ore must never exceed from 3.15 to 3.54 in. in thickness. Roasting is to be effected at a low temperature, and ought to proceed as far as the theoretical limit of one equivalent of sulphate, or two equivalents of oxide, for each equivalent of sulphide. After the first firing, which produces lead, and fresh roastings and firings twice or thrice repeated, the rich residues must be withdrawn from the reverberatory furnace, without having recourse to *ressuage* (reduction of rich residue in the same furnace immediately afterward), but rather by practicing this ressuage in a blast furnace."

The statement of Grüner, made in 1868, retains its value at present as regards the manner in which reverberatory work is to be carried on; it has become antiquated in its advocacy that non-quartzose ores ought always to be treated by the roasting and reaction method, because reverberatory treatment has been replaced almost wholly by blastroasting followed by blast furnace smelting.

The illustrations of the reverberatory furnaces given represent structures which can serve as models only in their general features. In a modern furnace the hearth would be built into an air-cooled iron pan supported by cross-bars resting on two longitudinal brick walls. This form would keep cool the most vulnerable part of the furnace, the bottom; it would draw attention to any leakage of lead, which is easily overlooked with a bottom built up solid from the foundation. The furnace of Couëron, France,[4] represents a somewhat cumbersome form of this idea. Further the fireplace in a modern furnace would be erected independently of the hearth, and thus allow for necessary expansion and contraction.

[1] *Loc. cit.*, p. 117.
[2] *Loc. cit.*, reprint, p. 511.
[3] "Lead," p. 491.
[4] Phillips, J. A.-Bauerman, H., "Elements of Metallurgy," Griffin & Co., London; Lippincott & Co., Philadelphia, 1887, p. 591.

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CHAPTER VIII

SMELTING LEAD ORES IN THE ORE-HEARTH

64. Introduction.[1]—This ancient method of treating high-grade galena ore has undergone many vicissitudes. At first the only method for working low-silica galena ores, it was largely replaced by the reverberatory furnace, and this, as has been shown, has seen its best days. Since the introduction of the bag house to collect fume, the ore-hearth has taken a new lease of life, so that in 1914 in Missouri[2] there were in operation 75 ore-hearths with an annual capacity of 230,000 tons galena concentrate. The bag house, however, is costly as regards installation and operation. The excellent results that have been obtained with the Cottrell electric process for the collection of fine dust and especially of fume point to the probability of its replacing filtration, as the cost both of installation and operation is low.

With the invention of the Newnam mechanical furnace (§66) the leading disadvantages of hard manual labor and of exposure to lead fumes have been so diminished that in the near future a larger output in lead than is the case at present may be expected.

The process carried on in the ore-hearth is mainly the roasting and reaction process. It resembles that in the reverberatory furnace, with this difference, that oxidation and reduction go on simultaneously, and that carbon in addition to sulphur acts as a reducing agent, the charge floating on a bath of lead. The PbO and $PbSO_4$, as soon as formed, react on undecomposed PbS; some PbO is reduced by C; and the liberated Pb trickles through the charge into the hearth bottom, overflowing into an outside kettle.

The same conditions are as necessary for the hearth treatment as for the reverberatory, with the exception that the Pb-content, usually 70 per cent., should not be less than 65 per cent. and the ore should be coarser.[3] The smallest permissible size is that of a pea, and nut-size is desirable. If fine ore is to be treated, it must first be pugged or agglomerated in a reverberatory furnace,[4] as in the ore-hearth it would be blown away. A hearth requires power and a blower, and much lead is volatilized; hence it is not suited for argentiferous ores. This rule, while true in a general way, has to be qualified by the fact that silver-

[1] Pulsifer, *Min. Eng. World*, 1914, XL, 501.
[2] Pulsifer, *Min. Eng. World*, 1914, XL, 1153.
[3] Dwight, *Eng. Min. J.*, 1916, CII, 677, 1026.
Garlichs, *op. cit.*, 948, 1026.
Choate, *op. cit.*, 1027.
[4] Middleton, *Eng. Min. J.*, 1905, LXXX, 10.

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bearing ores need not be excluded. The late W. W. Petraeus about 1895[1] did treat such ores and found that, while some silver was found in the dust, very little was detected in the fume.[2] With the recovery of dust in flues and of fume in bag houses, both of which are treated in the blast furnace, the loss in metal by dusting and volatilization is reduced.

The comparative loss in Pb at Raibl, in 1888,[3] where ore-hearth and reverberatory worked on the same ore, with imperfect means of condensation, was 10.6 to 7.3 per cent. The ore-hearth requires purer and higher grade ore than the reverberatory furnace, but consumes less fuel. According to Tunner,[4] a furnace similar to the one at Raibl consumed, per 100 lb. of galena, 7.31 cu. ft. of wood, while the ore-hearth required only 2.90 cu. ft. The ore-hearth has three times the capacity of the air furnace. The cost of treatment per ton of ore is the same with a single ore-hearth; with several running side by side it becomes less, as only one set of men is necessary to run the more powerful engine, and this consumes relatively less steam than the smaller one. That the ore-hearth cannot compete with the English or Silesian reverberatory furnaces as regards capacity and cost is clear. It has, however, one advantage over all reverberatories, that it is quickly started and stopped without much consumption of fuel (usually about 5 per cent. of the weight of the ore) or loss in heat, and thus serves its purpose in extracting at intervals from small amounts of non-argentiferous ore the major part of the lead. This is probably the reason why it found such favor in the Mississippi Valley, where small amounts of ore were often treated, and still are, by men who have mined it themselves.

The practice in the different ore-hearths is so very much the same that nothing from a general point of view need be said about it.

The products are similar to those of the reverberatory; there is, however, an intermediary product, a mixture of ore, slag, and fuel, called browse in England, which goes back to the charge in the ore-hearth. The gray slag produced contains much lead, and is smelted either in an ordinary blast furnace for lead, matte, and slag, or in a small blast furnace, known as the slag-eye furnace, if white paint is to be produced. The fume, which in former times used to be recovered imperfectly, is now collected in a bag house, sintered, and smelted either with gray slag and other lead-bearing materials in an ordinary blast-furnace,[5] or with gray slag alone in a slag-eye furnace for white paint and waste slag.

65. Influence of Foreign Matter.—It has already been said that the ore subjected to hearth treatment must be purer and richer than is necessary for the reverberatory furnace. This is because the foreign matter shows its bad

[1] Private communication.

[2] This plan of treatment has been patented by O. H. Picher, U. S. Patent No. 920-388, May 4, 1909; *Eng. Min. J.*, 1909, LXXXVIII, 256.

[3] *Oester. Zt. Berg. Hüttenw.*, 1888, XXXVI, 320.

[4] *Leoben. Jahrb.*, 1852, I, 262.

[5] Petraeus, C. V., U. S. Patent No. 568289, Aug. 17, 1897.

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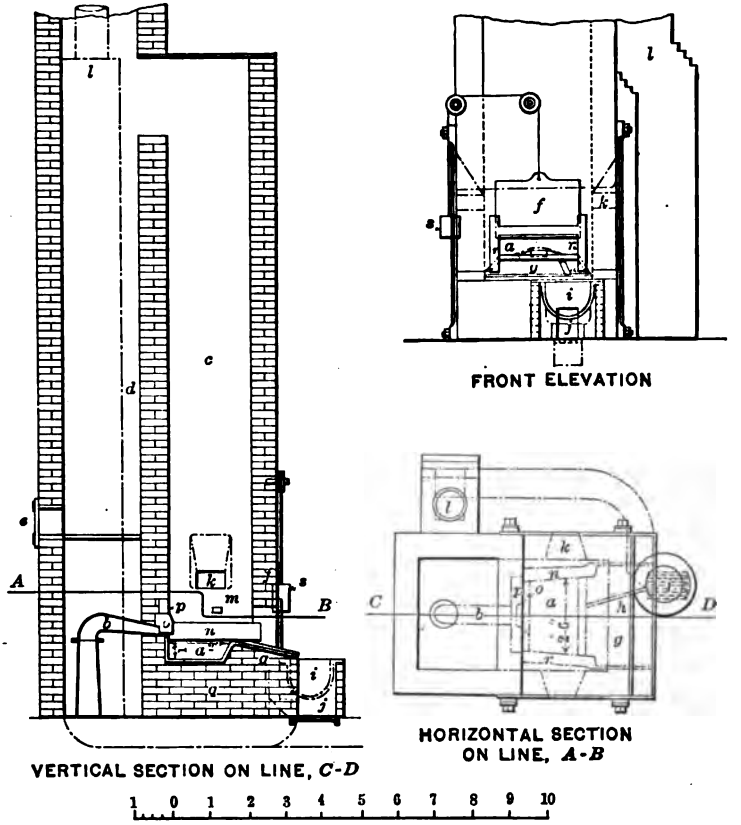
influence in a more marked degree. The chemical action is, however, the same as that described in §58.

66. Furnaces.—An ore-hearth, being a low fireplace surrounded by three walls with one or more tuyères at the back, cannot show much variety in construction or in manner of operating. However, seven different forms have been chosen by way of illustration. The Scotch, the Rossie, the Eagle-Picher, the Original Water-back, the Moffet, the Modern Water-back, and the Newnam.



FIGS. 68 to 70.—Scotch ore-hearth, Newcastle, England.

1. *The Scotch Ore-hearth.*[1]—Figs.[2] 68 to 70 represent the furnace used by Messrs. Cookson & Co. near Newcastle, in the North of England. The cast-iron hearth-box or well, *a*, which holds the lead, is set in brick-work, *q*. It is 2 ft. from front to back and about 2 ft. 6 in. wide; it is 1 ft. deep and holds about 2 tons of lead. In some furnaces the depth is only 6 in. and the capacity of

[1] Percy, "Lead," p. 278.
Sexton, *Eng. Min. J.*, 1895, LIX, 175.
Middleton, *op. cit.*, 1905, LXXX, 10.

[2] "Eighth Annual Report of the Local Government Board," 1878–79, supplement containing the Report of the Medical Officer for 1878, London, 1879, p. 281.

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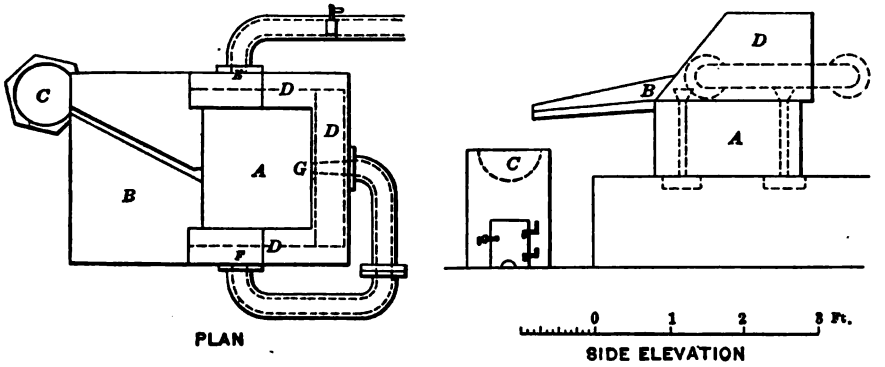
the well about 1,340 lb. of lead. The work-stone, *g*, an inclined plate, is cast in one piece with the hearth-box. It has a raised border on either side and at the lower edge, and a groove, *k*, which leads the overflowing lead toward the kettle, *i*, heated from a fireplace, *j*, below, the gases passing off through a flue into the chimney, *l*. On either side of the hearth-box and resting on it is a cast-iron block (bearer), *n*. Another cast-iron block, *o* (back-stone, pipe-stone), is placed at the back. It is perforated for the passage of tuyère *b*, which enters the furnace about 2 in. above the surface of the lead in the box. Upon *o* rests the upper back-stone, *p*, also of cast-iron. Thus lead and ore are entirely surrounded by cast-iron. The fore-stone, *m*, appears to be rather small. The brick shaft, *c*, carries off the fumes; at the back is a "blind flue" or pit, *d*, forming a sort of dust-chamber, to be cleaned through door *e*, the gases passing off upward. On the side of the shaft is the feed-door, *k*, for the introduction of fuel in front of the tuyère and the removal of any slag adhering to it, the charge being fed from the front. The hearth has at the front a shutter, *f*, sliding in a grooved frame, *r*; it is raised and lowered by means of counterpoise *s*. Peat, formerly used as fuel, is now replaced by bituminous coal.

It is characteristic of the method of working in the Scotch hearth that the process is discontinuous. After smelting from 12 to 15 hr. the hearth becomes too hot and must be allowed to cool for about 5 hr. before work can be started again.



FIGS. 71 and 72.—Rossie or air-cooled ore-hearth.

2. *The Rossie or Air-cooled Ore-hearth.*[1]—This hearth, shown in Figs. 71–72, was formerly in use at Rossie, N. Y., with wood as fuel; it is in operation in the Mississippi Valley in a modified form. The cast-iron hearth-box, *A*, is 24 by 24 in. and 12 in. deep, and the casting 2 in. thick. The cast-iron work-stone, *B*, is 32 in. wide and 22 in. from front to back; it has flanges on the sides, slopes 1 in. in 12 in., and is provided with a diagonal groove in which the lead flows from the hearth to the cast-iron kettle, *C*, set over a fireplace. The cast-iron air jacket, *D*, is 14 in. high and 6 in. wide; with the casting about 0.75 in.

[1] Anon., *Am. J. Sc. and Arts*, 1842, XLII, 169.
Percy, "Lead," p. 289.

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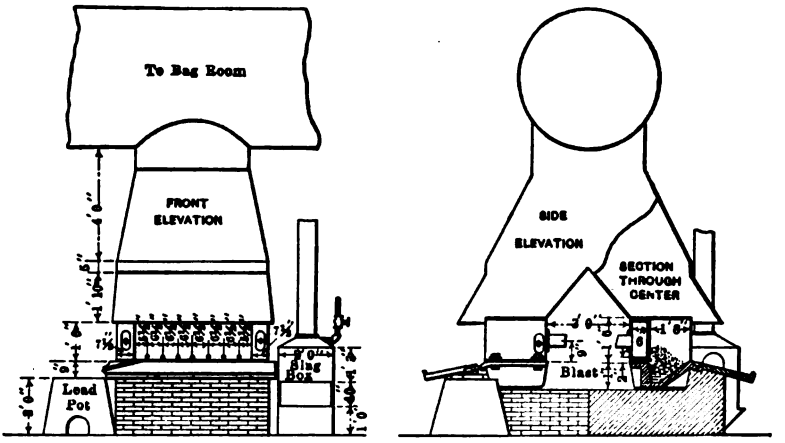
thick, there is left an air-space 12.5 by 4.5 in. The blast enters at *E*, passes through *D*, leaves at *F*, and is delivered to the tuyère opening, *G*, 2 in. above the top of *A*. By this arrangement the sides and back of the hearth-walls are air-cooled, and the blast is heated.

The characteristic of the mode of operating was that the hearth could be worked continuously; hence its smelting power was greater than that of the Scotch hearth, but the hot blast caused greater loss of lead by volatilization. For this reason the hearth was abandoned. Data are assembled in Table 32. In our days all the fumes are recovered; the heating of the blast by this simple means is not out of place as is seen by its reintroduction at Granby and Webb City, Mo., and Galena, Kans. The furnace was in operation in 1906 at several Austrian smelteries.[1] Thus, Gailitz had six furnaces built around a central chimney, Scherian has two similarly placed, and Kreuth two. Wet condensation of fumes is common at these plants.



FIGS. 73 and 74.—Eagle Picher Lead Company ore-hearth.

3. *The Eagle-Picher Lead Co. Ore-hearth.*—This modern form of air-cooled ore-hearth is shown in Figs. 73–74. The hearth, 5 ft. 5 in. long and 1 ft. 8 in. wide, has air-cooled sides and back; the blast is thereby heated for the tuyères. The lead-pot is placed on one side, the slag-box, provided with a sprinkler on the other. Furnaces are set back to back with an intervening space 3 ft. wide. The hoods which carry off dust, vapor, and gas end in a main flue which is provided with hoppers placed between pairs of furnaces. From them dust is drawn off into carts. The fume is collected in a bag house which contains for every furnace 280 bags 18 in. in diameter and 32 ft. long. The blue powder collected by the bags is burnt in the usual way and smelted with gray slag in a slag-eye furnace for sublimed lead paint. The blast pressure in the furnace is 7 oz. per square inch and over.

[1] Government Report on "Bleivergiftungen, etc.," Hölder, Vienna, 1905; abstract, *Min. Ind.*, 1906, XV, 533.

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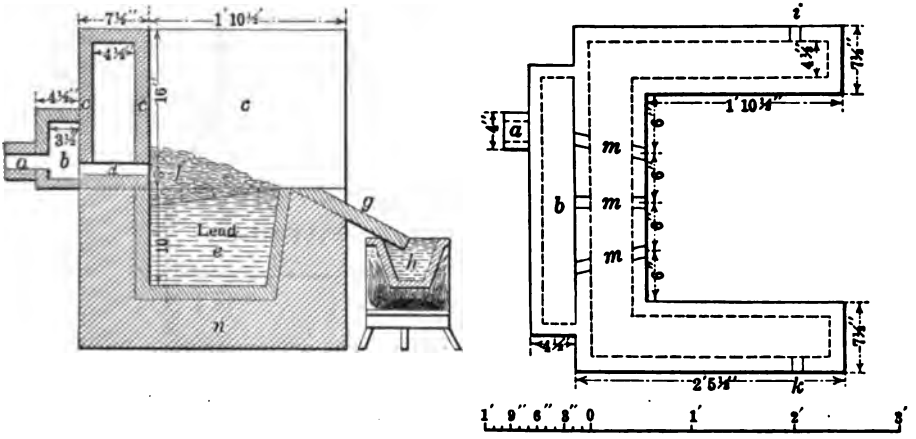
In 24 hr. the ore-hearth treats 28,000 lb. lead ore assaying 75 per cent. Pb; requires 12,000 lb. coal and 4000 lb. lime; and produces 1200 lb. metallic lead, 4,200 lb. gray slag, 2240 lb. flue-dust, and 5040 lb. fume. Data of the ore-hearth are assembled in Table 32.

4. *The Original American Water-back Ore-hearth.*[1]—Figs. 75 and 75*a*[2] show the larger sized furnace with three tuyère-holes, *m*, at the back, where the older, smaller form had only one. The hearth-box, *e* (filled with lead, the charge of ore and charcoal floating on top) is set in brickwork, *n*. It holds about 2500 lb. of lead. The work-stone, *g*, leading to the kettle, *h*, forms a separate casting from the hearth-box. The three sides of the furnace are formed by a



FIGS. 75 and 75*a*.—Original American water-back ore hearth.

water-cooled cast-iron jacket, *cc*, 1½ in. thick, called a tuyère-plate, resting on the hearth-box. The water enters at *i*, and passes out at *k*. At the back of the tuyère-plate is the wind-box, *b*. The blast enters this at *a*, and passes through three wrought-iron tuyère nozzles, *d* (from 1 to 1½ in. in diameter), in tuyère holes, *m*, into the hearth, at from 1 to 3 in. above the level of the lead. The hood placed over the furnace to carry off the fumes and gases is not shown in the drawings.

The work in the American ore-hearth is continuous, as distinguished from the Scotch hearth. This is made possible by water-cooling the sides of the furnace, an arrangement which protects the castings and the tuyères. The fuel used is wood, charcoal, and bituminous coal. Data are assembled in Table 32.

5. *The Moffet Ore-hearth*[3] (Figs. 76 to 79).—In Figs. 76 and 77 the

[1] Williams, "Industrial Report, Geological Survey of Missouri," 1877, p. 36; *Tr. A. I. M. E.*, 1876–77, v, 324.

[2] Broadhead, "Geological Survey of Missouri," 1873–74, p. 492.

[3] Dewey, *Tr. A. I. M. E.*, 1889–90, XVIII, 674.
Clerc, *Eng. Min. J.* 1885, XL, 4.
Ramsay, *Scientific American Supplement*, May 14, 1887, No. 593.
Holibaugh, "Lead and Zinc in Missouri," New York, 1895, p. 37.

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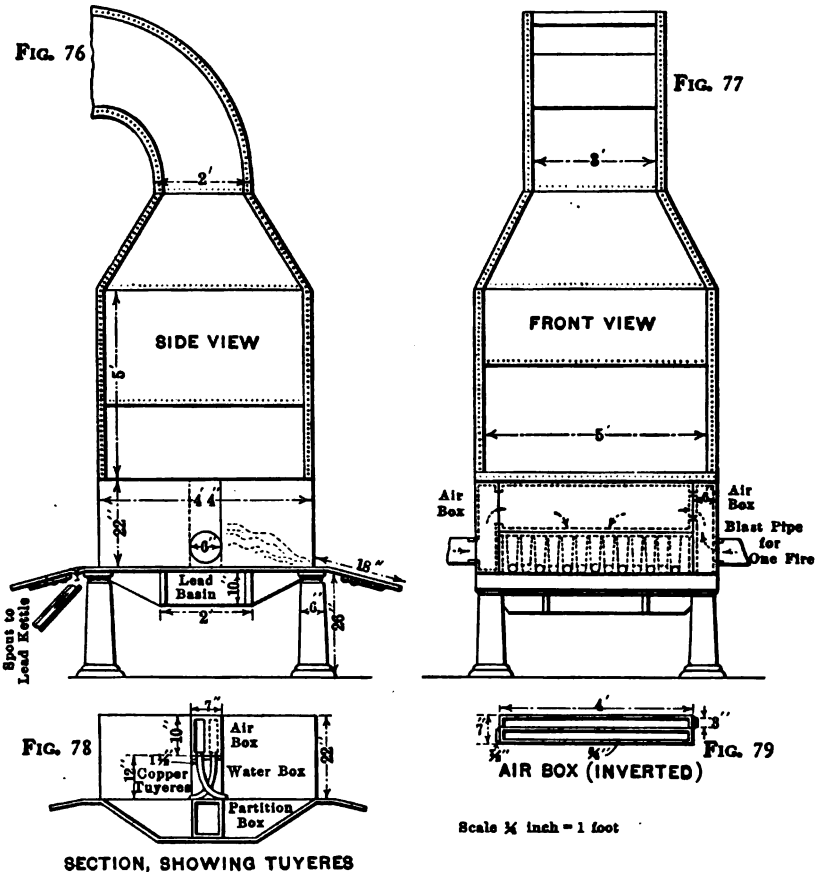
furnace is seen to rest on four pillars. The hearth-box (lead basin) is not set in brickwork, and the lead is thus kept cool. Two furnaces are set back to back, the fumes passing off under one hood. The lead runs through a separate spout near the top of the work-stone into a cast-iron kettle 31 in. in diameter and 44 in. deep, not shown in the drawings. It is set in a cast-iron cylinder and heated from below, one cord of wood a week being required for the purpose. The two furnaces work independently. On the floor of the lead basin rests the



FIGS. 76 to 79.—Moffet ore-hearth.

partition-box, having an opening near the bottom for the lead to enter. It serves as a support for the water-box, which cools the hottest part of the furnace, and upon which rests the air-box, consisting of two separate chambers (Fig. 78), where the heated blast passes down through the water-box by means of fourteen 1-in. copper tuyère pipes, seven on either side. The working opening is 15 in. high. A No. 5 Baker blower furnishes the blast. The fuel used is bituminous coal.

The combined use of water- and air-cooling is to protect the casting in con-

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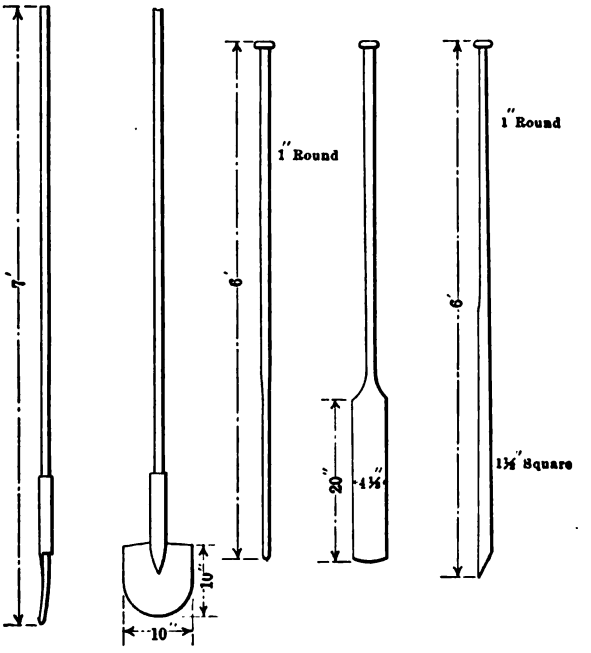
tact with the charge and to preheat the blast. The hot-blast saves fuel and increases smelting power, but it also increases volatilization of lead. The latter is of no detriment as long as the fume is to be converted into white paint, the lead-content of which brings a higher price than when in the metallic state.

The double ore-hearth, which is compact and saves space, piping, etc., has not found much favor, as the work of the front man interferes with that of the back man. At present most furnaces are single and are placed in a single row whenever this is possible; a double row, however, has its advocates.



FIGS. 80–82.—Tools used with the ore-hearth.

The tools, a long-handled round-pointed shovel, a paddle (spud), and a square bar (poker) are shown in Figs. 80 to 82. There are further required a ladle and a skimmer for the lead. With the Scotch ore-hearth[1] the round-pointed shovel is used at some works for ore only, and a square-pointed one for working in the furnace; then a short scraper is introduced through the feed-door to remove slag adhering to the tuyère nozzle.

6. *The Modern American Water-back Ore-hearth.*—The characteristics of this furnace are, that the cast-iron hearth-box is supported by short columns or a cast-iron frame in order to air-cool the lead; that the upper sides are two water-cooled castings; that the inside working length does not exceed 4 ft.; and that there are provided two hoods, an inner and smaller for carrying off dust

[1] Percy, "Lead," p. 282.

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and the bulk of the fume by means of a suction fan, and an outer for removing the remaining fume by natural draft. Such a furnace is shown in Fig. 98.

7. *The Newnam Mechanical Ore-hearth.*[1]—The idea of replacing the arduous and unhealthy hand-labor at the ore-hearth by some mechanical device has been held by many operators. For example Brown[2] patented[3] a device which, however, did come into practical use. The only successful apparatus so far is that of Newnam constructed at the works of the St. Louis Smelting & Refining Co., Collinsville, Ill., put into operation there, and since adopted by other plants. It is shown in Figs. 83–97. Figs. 83 and 84 give a front and side elevation of the furnace; Figs. 85 and 86 horizontal and vertical sections of the lead basin; Figs. 87–89 elevations and sections of the middle water-jacket; Figs. 90–92 elevations of the right and left end-jackets and section of the right end-jacket; Figs. 93–95 details of the cast-iron stand supporting the lead basin; and Figs. 96–97 perspective views of the furnace with rabbling mechanism at beginning and end of trip.

The basin, *a*, Figs. 85 and 86, is 8 ft. long, 19.5 in. wide at the top and 10 in. at the bottom, and 10 in. deep; it has a vertical back and a sloping front. The sides of the furnace are cast-iron water-jackets provided with lugs which fit into similar projections of the basin. There is a middle jacket, *b*, Figs. 87–89, 3 ft. 11.5 in. long and 18 in. high, which has a water-space, 4 in. wide, and eight tuyères, 1⅝ in. in diameter. The right and left side-jackets, *c*, Figs. 90 and 91, are cast in one with the end-jackets, *d*, shown in elevation in Fig. 92. The right and left jackets each have four tuyères, making for the furnace a total of 12 tuyères. From the basin, *a*, Fig. 85, the lead is discharged into the molds of a molding car, Fig. 96, through the siphon tap, *e*, Fig. 85.

The inner and outer hoods, Figs. 83 and 84, are the same as those to be discussed in connection with Fig. 98. Details of the cast-iron frame supporting the lead basin are clearly shown in Figs. 93–95.

The furnace with the mechanical rabbling device is represented by the perspective illustrations, Figs. 96 and 97, because a comprehension of the details requires a large number of drawings which are out of place here. The rabbling machine is hung from an overhead traveling carriage, 3 ft. long with a gauge of 3 ft. 10 in. The steel frame supporting the carriage is 14 ft. 6 in. by 4 ft. 1.75 in. and 7 ft. 8 in. high. Assuming that the carriage is at rest at the right end of the hearth, Fig. 96, a lever is pulled to start the machine on its path to the other end during which the rabble-arm is given a motion similar to that of the hand-rabble. When the rabble-arm is withdrawn from the fire, the crariage moves forward a distance of 4 in., and starts the arm on the next stroke. When the carriage has arrived at the opposite end of the furnace, Fig. 97, it is stopped automatically, the rabble-arm is withdrawn, and the carriage returned to its original position. In Fig. 97 is shown a 1-h.p. motor which is connected through gearing with the main shaft. This carries at its left extremity a

[1] *Tr. A. I. M. E.*, 1916, LIV, 485. Private Notes, July, 1916.
[2] *Tr. A. I. M. E.*, 1911, XLII, 402.
[3] U. S. Patent No. 888582, May 26, 1908.

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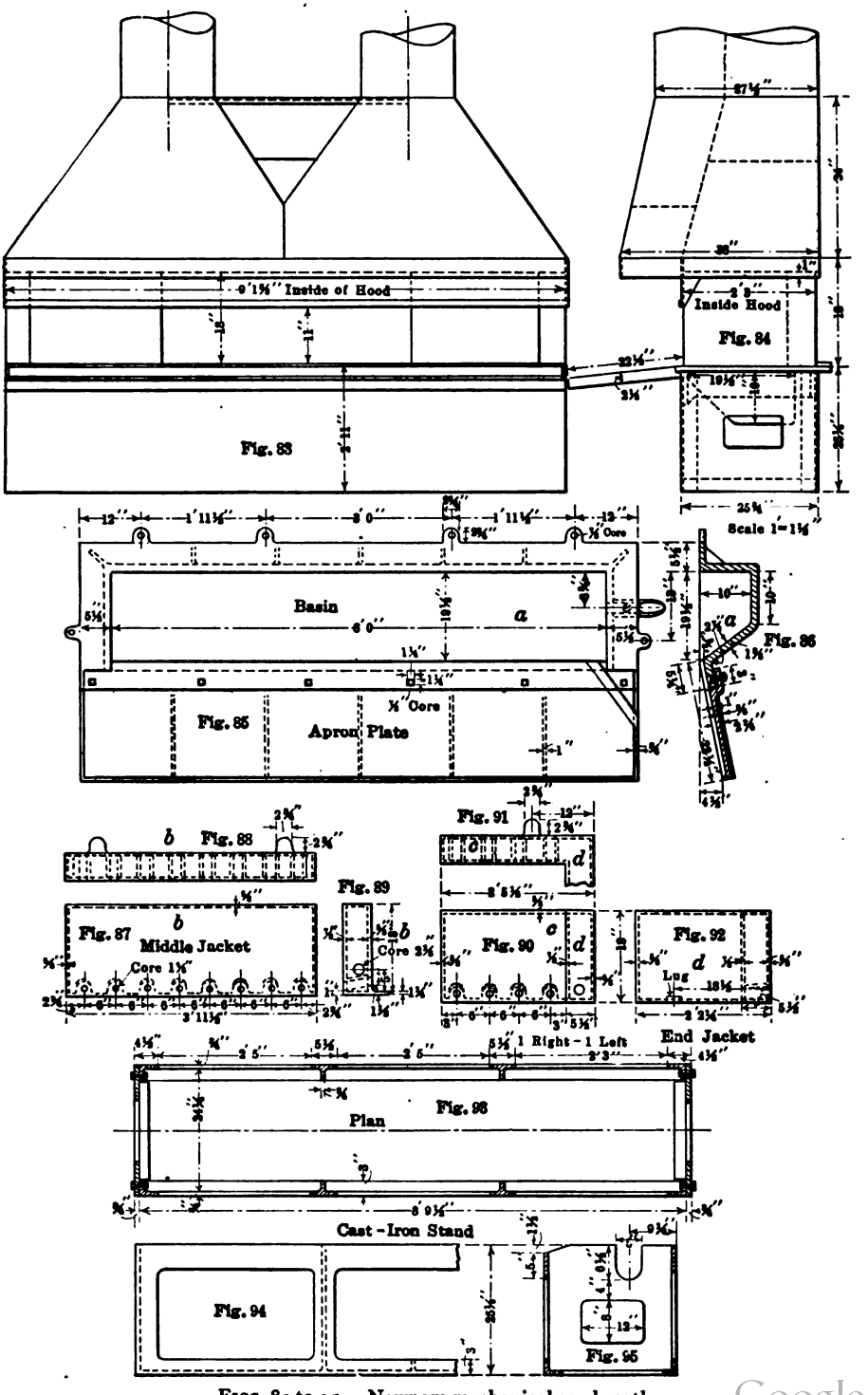

FIGS. 83 to 95.—Newnam mechanical ore-hearth.

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crank-pulley to which is attached the rabble-arm; the latter is supported by a rest and thrown in or out of action by means of a tooth-clutch. The main shaft carries a sprocket-wheel which rotates by means of a chain a counter-shaft in

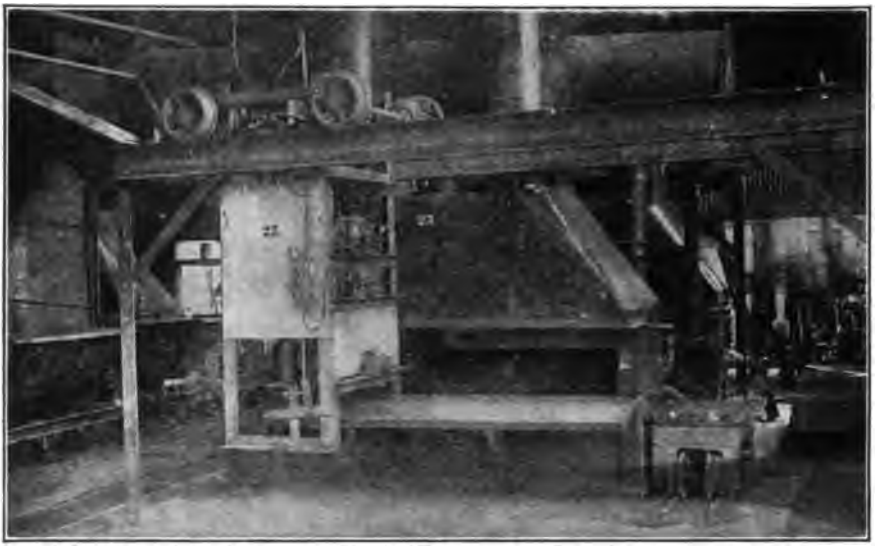

FIG. 96.—Rabbling machine at beginning of trip.

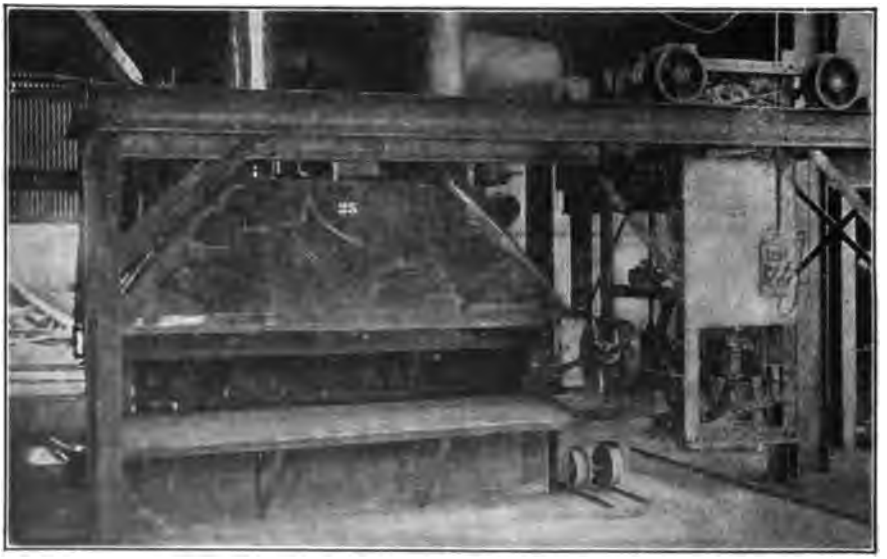

FIG. 97.—Rabbling machine at end of trip.

the carriage; the latter has a toothed clutch and drives by means of a bevel gear, Fig. 97, one axle of the carriage when this is to move from right to left. The main shaft carries also an excentric, Fig. 96, which is connected through three levers (one straight- and two angle-) with a ratchet and pawl to drive the

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other axle of the carriage when it is to travel from left to right. The motion of the carriage is automatically stopped and reversed at either end of its trip.

While the mechanical arm rabbles the charge, one man (the helper) follows it, pushes back loose charge with a long-handle shovel, and picks out gray slag and delivers it on to the apron plate. He is followed by a second man (the charger) who transfers the gray slag to the water-box, spreads a thin layer of ore on to the charge as fast as it is shoveled by the helper, and adds coke-breeze as needed. By the time the trip down the hearth has been completed, the ore first charged is ready to be rabbled.

In front of the hearth is suspended a Wiegand chain screen[1] to protect the men from heat. It consists of a number of strands of steel chains suspended closely from a bar so as to form a curtain similar to a Japanese screen. Data comparing the work with the hand-hearth are given below.

67. Mode of Operating.—With a new hearth a fire is first kindled with wood; then coal is added, and the blast started. This will soon set the fuel in a blaze; more coal is then added, ashes and clinkers are removed, and in a short time a body of glowing fuel is obtained, filling the entire hearth. Some residue from a previous run is then spread over the back part of the fire and the first charge, from 10 to 20 lb. of ore, given. This soon becomes red hot, and the lead that is set free trickles through the body of the fuel and collects on the bottom. More ore is added. The contents of the hearth are then lifted up with the poker and kept open in order that the heat may be distributed through the entire mass. Parts that have become melted and form lumps are drawn out with the shovel or spud on to the work-stone, the slag is separated from rich residue and thrown into a water-box, while the residue is returned to the furnace. Ore and fuel are again added, and the operation continued until the hearth is full of lead.

If, in the regular course of operating, the work on a hearth is to be stopped, the charge floating on the lead is raked off, and the lead in the hearth-box allowed to solidify. In working such a furnace again, the lead is melted in about 1 hr. by starting a fire with wood or charcoal, and then keeping it going with coal.

The hearth is now in normal working order; it is filled with lead, and glowing fuel, mixed with partly fused and partly reduced ore, floats on top of it, as seen in Fig. 75.

With the Scotch ore-hearth from 12 to 30 lb. of ore are mixed with 1½ or 2 per cent. of lime, spread over the glowing floating mass, and then covered with a little fuel. The charge is now exposed for from 3 to 5 min. to the action of heat and blast, and the blast shut off. After this, one of the furnace-men inserts the poker at different places into the lead, loosens and stirs the charge, and raises it slowly, for which purpose the spud is sometimes used. The other man draws the semi-fused mass out from below with the shovel upon the work-stone; this allows the ore on the surface to sink down. What has been drawn upon the work-stone is broken up and the slag separated from the half-decom-

[1] *J. Ind. Eng. Chem.*, 1916, VIII, 836.

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posed ore. The former is thrown aside (sometimes into a water-box); the latter goes back to the furnace, slaked lime being spread over it, if necessary. Any slag adhering to the tuyères is then removed; some fuel is distributed in front of them and over the charge; fresh ore is spread on the fuel, and this again covered with fine fuel; then all is ready for a second operation. As the smelting proceeds, the bulk of the lead that is set free trickles through the charge into the hearth-box and overflows through the groove in the work-stone into the kettle. Here it is sometimes poled before being ladled, syphoned, or drawn off through a spout into molds. Some lead passes off with the fumes and the rest goes into the slag.

Pattinson[1] calls attention to the following points in managing the ore-hearth. The amount of blast and its distribution through the entire charge should be carefully regulated, the half-reduced ore should be exposed on the work-stone to the oxidizing action of the air, and the additions of lime and fuel judiciously made.

With American ore-hearths the blast is not shut off every 3 to 5 min., but works continuously, and the process is therefore not interrupted. The first step in normal work is to spread a few shovelfuls of coal over the fire, and then five or six shovelfuls of ore mixed with from 0.5 to 1.0 per cent. lime. The fire is left undisturbed for from 1 to 1.5 min., during which time the charge is more or less oxidized, the flame breaks through, and the surface of the charge becomes incrusted. The poker is thrust by the charger over the edge of the basin into the lead, and the slightly caked mass lifted and loosened. He begins at one, say the right, end of the furnace and proceeds slowly to the left. At the same time the helper, following him, picks out with a shovel glowing fuming lumps (gray slag) that have been formed, places them on the apron, and then tosses them into the water-box. The helper now exchanges the shovel for the paddle and loosens the fire a second time, while the charger follows him and picks out newly formed slag with the shovel. Lastly the paddle is laid aside, and the shovel used to remove most of the remaining slag. When this is done, new coal and new charge are fed. The whole operation lasts about 2 min. The men rest from 1 to 1.5 min., during which time the charges become partly oxidized and caked.

The charge is made up of either galena alone or a mixture with sintered bag-house fume. Formerly slaked lime was used exclusively as flux; with the replacement of bituminous coal by coke-breeze, the greater heat generated by the combustion has permitted the use of limestone, which works satisfactorily under the new conditions. The amount of fuel used ranges from 4 to 8 per cent. of the ore mixture.

The work is very exacting, as it is essential for success that the charge remain open in order that the blast may easily penetrate all parts. Usually there are employed two men to a hearth on an 8-hr. shift; at Granby there work four men on a single shift of 9 hr., one pair relieving the other at short intervals. The better the stirring, the larger is the smelting power of a furnace, and the greater its production. In most plants the amount of metallic lead produced

[1] Percy, "Lead," p. 288.

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by a crew during its shift in summer is lower than that in winter. Thus, from the normal amount of 7,000 lb. ore to be worked per shift of 8 hr., the men extract 50 per cent. metallic lead in summer, while in winter the figure may reach 60 per cent.

The lead set free in smelting sinks through the fire and collects in the basin. In order to make it overflow freely through the channel into the collecting-pot, the shovel is forced through the charge into the lead and then moved up and down, a pumping motion which raises the lead and causes it to flow readily into the channel. In the pot the lead is drossed and then cast into 80- or 100-lb. bars; the dross goes back to the furnace. The lead is cleaned by liquating and poling, and is then cast into market bars, unless it is to be desilverized as at Collinsville and Federal, Ill.

The mode of operating with the Newnam mechanical ore-hearth is the same as that in a hand-worked hearth except that the stirring is done by machinery as shown in §66.[7] In a comparative test of 4 weeks' duration with a galena concentrate assaying Pb 72.5 and S 15.1 per cent., the Newnam 8-ft. ore-hearth treated 13,179 lb. ore, produced 6443 lb. pig lead and 3318 lb. gray slag (Pb 43.7, FeO 12.8, CaO 9.9, S 1.9, Insol 12.6 per cent.), used 3.6 per cent. coke-breeze, and made 37.2 rabble-trips per hour. The figures for the 4-ft. hand ore-hearth were; ore 5091 lb., pig lead 2030 lb., gray slag 1329 lb. (Pb 45.0, FeO 12.2, CaO 9.0, S 2.6, Insol 12.2 per cent.), coke-breeze 8.8 per cent. Of 100 lb. lead in the ore, the Newnam hearth recovered 67.44 as pig lead, 15.18 as gray slag, and 17.38 as dust and fume; with the hand ore-hearth the figures were, pig lead 55.00, gray slag 16.20, dust and fume 28.20. In the gray slag produced by the parallel tests, the lead was present as $PbSO_4$ 10.0, PbS 6, PbO 25.0, Pb 8.5, total 49.5 per cent. Dust and fume were formed in the proportions of 1 lb. dust to 2 lb. fume. The dust assayed Pb 62.0 and S 11.1 per cent.; the lead in it was distributed as $PbSO_4$ 18.9, PbS 55.4, PbO 2.0 per cent. The burnt fume assayed, Pb 76.0 and S 5.9 per cent., the lead being distributed as $PbSO_4$ 53.6, PbS 1.2, and PbO 41.5 per cent. The S-elimination with the Newnam hearth was 87.9 per cent., with the hand hearth 80.6 per cent.

Treatment of a galena concentrate with Pb 82.0 and S 11.2 per cent. gave per 8-hr. shift the following figures, ore 14,436 lb., pig lead 10,790 lb., gray slag 1075 lb., coke-breeze 2.4 per cent., crushed limestone 2.0 per cent. The percentage of lead recovered in the products was, pig lead 91.15, gray slag 4.25, dust and fume 4.60 per cent.; the S-elimination was 94.7 per cent.

Smelting flue-dust (Pb 62 per cent.), pugged with lime, and burnt bag-house fume (Pb. 76 per cent.), and mixed to contain 76 per cent. Pb, and using 10 per cent. coke-breeze to make up for the lack of sulphur, gave as pig lead 81.3 per cent. of the Pb charged, as gray slag (Pb 32.4, FeO 14.7, CaO 11.1, S 1.2, Insol. 18.7 per cent.), 10 per cent., and as fume 8.7 per cent.

The yield in pig lead and the elimination of sulphur are greater in the Newnam than in the hand hearth, the work is easier and less unwholesome, and the cost a great deal less.

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68. Plants.—In a modern plant the hearths usually stand separately in a single row in the hearth-building. In front and back are tracks, on the front track arrive concentrates, fuels, and fluxes to be emptied into wooden boxes or bins; on the back track are taken away on trucks the bars of lead, and in cars the gray slag. Back of a row of furnaces runs the blast main from which branch-pipes deliver the air to the furnaces. On the left side of a furnace is a branch-track for the slag-car, and on the right the lead-pot and frame to support the molds. Each hearth has a small inner hood which rests on the side-jackets and is connected with a fan, and a large outer hood which carries off by natural draft the fumes, arising from the gray slag, and other gases. The stove-pipe which forms the chimney for the fireplace of the lead-pot extends into the outer hood and thus furnishes the necessary draft.

The gases from a row of furnaces pass either into a dust-chamber and thence into the main flue, or go direct into the main flue, which serves as dust-chamber. They travel then through a series of inverted U-shaped pipes (goose-necks), which cool the gases and cause the larger part of the dust to settle, and enter the suction fan. This draws the gases from the furnaces through the cooling and settling devices mentioned and delivers them to the bag house, where the remaining dust and all the fume are filtered. In plants which smelt the gray slag from the ore-hearth with the sintered bag-house product in an ordinary blast furnace for lead, matte, and slag, the gases from the blast furnace usually enter the main flue of the ore-hearth near the fan, and are filtered By this manner of operating, the losses in lead have been reduced to a tota of 2 per cent.

Table 31 by Pulsifer[1] gives the leading facts of the condensation plants of five modern ore-hearth smelteries.

TABLE 31.—COLLECTION OF ORE-HEARTH FUME

	Federal, Ill., 1901	Galena, Kans., 1902	Collinsville, Ill., 1903	Granby, Mo., 1904	Webb City, Mo., 1910
Flue, ft	1,794	800	600	950	1,150
Serves	Hearths	Hearths	Hearths and blast furnace.	Hearths	Hearths and cupola
Fan, kind	Buffalo	Buffalo	.....	Am. Blower Co.	Buffalo
Fan, diam., in.	134×68	90×<	196	96×36	100
Bag house material above brick cellar	Brick	Wood	Brick	Sheet iron	Wood
Bag house, size, ft.	60×150	68×100	38×78	42×90	56×75
Bag house, compartments, No.	1	1	4	1	1
Bag house, cellars	6	.....	20	8	5
Bags, No	1,614	680	2,160	640	720
Bags, material	Cotton	Cotton	Cotton	Cotton	Cotton
Bags, size, in. × ft.	18×26	18×30	18×36	18×30	18×25

[1] *Min. Eng. World*, 1914, XL, 502.

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Some details of these plants are the following:

1. *Federal, Ill.*[1]—The ore treated is a galena concentrate averaging Pb 70, CaO and MgO 4, SiO_2 2, Cu 0.15, S 13, Zn 2.5, FeO 5 per cent. There are in operation 30 ore-hearths placed in a row, and 16 ft. apart. An ore-hearth is 4 ft. long; has a lead basin 10 in. deep, 13.5 in. wide at the bottom, and 18⅞ in. at the top; and a cast-iron work-stone 16⅞ in. wide. On top of the basin rests the cast-iron water-jacket, which is 5 ft. long, 2 ft. 4 in. wide, 1 ft. 4 in. high, and consists of two sections joined at the back by lugs; the castings are 1 in. thick and have a 4-in. water-space. The back-jackets have eight 1-in. tuyère-holes 2 in. above the basin. Air is supplied from a 5-in. blast-pipe connected with a 12-in. main. On the jackets rests a small inner hood, and a larger one is suspended from the structure of the building. To the right of the hearth is the coal-fired lead-pot. The fuel used is coke. Two men work in an 8-hr. shift. A hearth treats 7000 lb. ore in a shift. There are produced in winter 24 and in summer 18 100-lb. bars of lead in a shift; the yield in lead as metal averages 50 per cent.; the coke consumption is about 8 per cent. of the weight of the ore. The ore-hearth and blast furnace leads are freed from drosses in two liquating reverberatory furnaces with brasqued hearths, 6 ft. 2 in. by 7 ft. 8 in., and fire-boxes 3 ft. 9 in. by 2 ft. 7 in. The lead runs into a 40-ton kettle to be poled with air, skimmed, and then siphoned into molds placed on a horizontal rotating casting table.

2. *Galena, Kans.*[2]—The ore, a mixture of sulphide and oxide lead ore, with from 2 to 10 per cent. blende, assaying about 70 per cent. Pb, is smelted in four ore-hearths, 5 ft. long, having air-jackets. The fuel used is coal; lime is added to the charge. Two men treat 7000 lb. charge in an 8-hr. shift; from 40 to 60 per cent. of the lead is recovered as metal. The lead from the kettle near the hearth is ladled into a two-wheel pot, transferred to a 4.5-ton cast-iron kettle, with discharge-spout at the bottom closed by a cock, poled with faggots, and molded. (See also Tables 31 and 32.)

The works were destroyed by fire in 1916, and are to be rebuilt with Newnam hearths.

3. *Collinsville, Ill.*[3]—The galena concentrate smelted assays Pb 68, CaO and MgO 3.0, Insol. 4.0, Cu 0.1, S 14.0, Fe 4 per cent. and Ag 1 oz. per ton. Up to 1915 there were in operation 24 ore-hearths, similar to those in use at Federal (see above), but placed in two rows back to back with flue between, which ended in a horizontal main. These hand-hearths have been replaced by 16 Newnam mechanical hearths (§66[7]).

The leading features are given in cross-section in Fig. 98. This shows two rows of ore-hearths, *a*. With the hearth to the right is seen the cast-iron frame, *b*, which holds the molding kettle and has a fireplace for keeping the lead liquid; the fire-gases pass off through a stove-pipe, *c*, ending in the stack of the outer hood, *d*. Between the two rows of furnaces is placed the blast-pipe, *e*,

[1] Pulsifer, *Min. Eng. World*, 1913, XXXIX, 375.
[2] Brinsmade, *Min. Eng. World*, 1909, XXXI, 1029.
[3] Pulsifer, *Min. Eng. World*, 1913, XXXIX, 681.

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from which branches, *f* and *f'*, deliver to the furnaces. The water-main, *g*, with branches, *g'* and *g''*, furnishes cooling-water to the jackets. The gas-main, *h*, with gas inlet-pipes, *i* and *i'*, delivers dust through outlets, *j*, into hoppers, *k*, provided with gates, *l*, from which it is discharged into cars, *n*. A hearth has two hoods. The inner hood, *o*, placed directly over the hearth, carries dust,

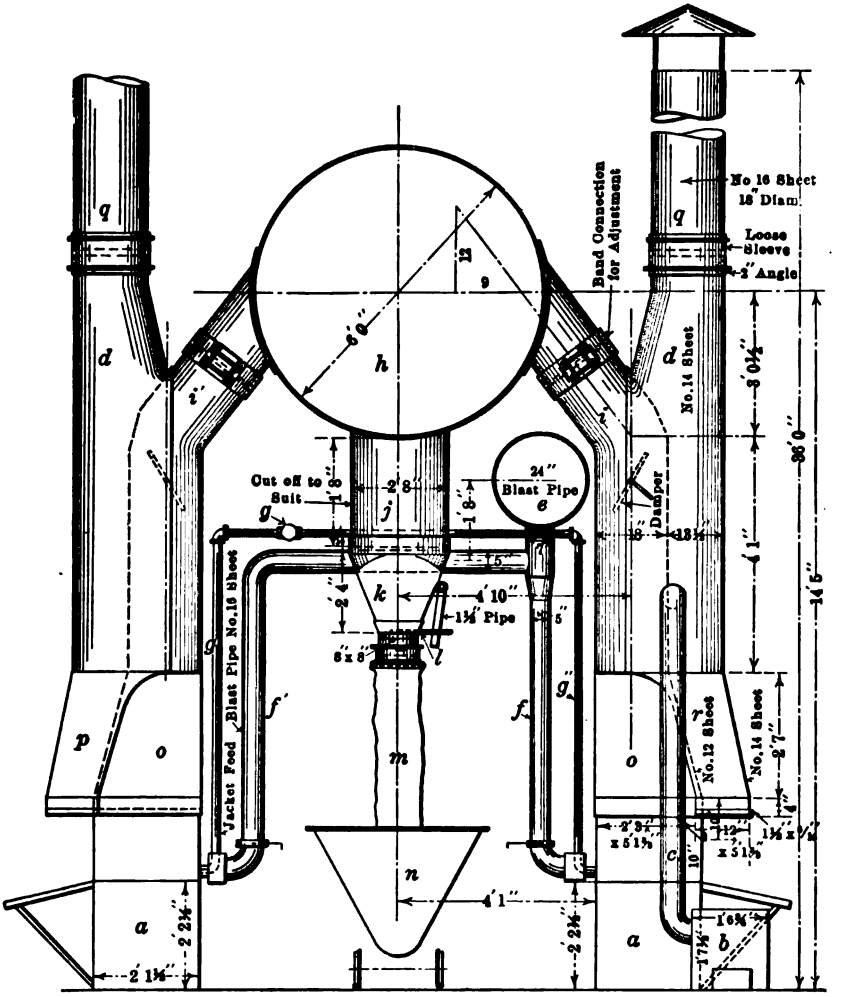

FIG. 98.—Hand ore-hearth, Collinsville, Ill.

fume, and gas through pipe, *i*, into gas-main, *h*; the outer hood, *p*, attached to the inner, draws stray fumes away from the furnace-men into the open through a sheet-iron stack, *g*, 36 ft. high measured from the furnace-floor.

Two men work in an 8-hr. shift. Coke to the extent of 4 to 5 per cent. serves as fuel; no lime is used. A hearth treats in 8 hr. 7000 lb. concentrate and produces 6200 lb. pig lead, which is approximately 55 per cent.; of the remainder,

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16 per cent. goes into gray slag, and the rest (29 per cent.) into dust and fume collected in the bag house.

The ore-hearth and blast furnace leads were formerly softened together for 12 hr. in a reverberatory furnace with hearth 18 by 11 ft. and 16 in. deep, holding 120 tons lead, and fire-box 8 by 12 ft.; the firebridge was water-cooled. There were produced 2.5 per cent. dross and 1 per cent. skimming. At present the lead is cleaned in two 5-ton kettles and desilverized by the Parkes process (§231) in four 60-ton kettles. The desilverized lead is freed from Zn in 12 hr. in two reverberatory furnaces (hearths, 10 by 13 ft.; fire-boxes, 2 by 7 ft.) each holding 55 tons, tapped into corresponding poling kettles, and siphoned into molds resting on a casting-wheel. (See also Tables 31 and 32.)

4. *Granby, Mo.*[1]—The ore, a galena concentrate mixed with sintered fume, is smelted in five ore-hearths, 4 ft. long, 15 in. wide, and 10 in. deep, provided with water-jackets 12 in. high. A furnace has four tuyères pointing downward; the apron plate is 18 in. wide. The furnaces are worked only during the day, from 6 a. m. to 4 p. m.; four men tend a furnace, two working at a time for 20 min. At 6 a. m. a charcoal fire is made on the frozen lead in the basin, and coal added; at 7 o'clock the lead is liquefied and heated. Now ore mixed with lime is spread over the glowing fire and worked in the usual way, only with more vigor and more coal. Air-blast on the floor and higher up serves to keep the men cool. The blast pressure for the furnace is from 6 to 8 oz.; the higher pressure is used with oxide ore prevailing. A furnace treats during the day 14,000 lb. charge, requires 2 bu. lime, consumes — lb. coal, extracts 75 per cent. of the lead as metal, makes 2000 lb. gray slag with 40 per cent. Pb (=10 per cent. of the lead of the charge) which is sold, and 480 lb. fume with 55–60 per cent. Pb (=15 per cent. of the lead of the charge). (See also Tables 31 and 32.)

5. *Webb City, Mo.*[2]—The ore, a galena concentrate with 80 per cent. Pb, is treated in six air-cooled ore-hearths standing in a row, each covered by a single hood ending in a pipe provided with a damper. The pipes lead into a main connected with a fan. A hearth is 60 in. long, 18 in. wide, and 7 in. deep. The fuel used is coal. Two men tend a furnace in a 6-hr. shift. Between shifts the lead is allowed to freeze. Each shift treats 7000 lb. ore mixed with 400 lb. lime. Of the lead in the ore, 65 per cent. is recovered in the hearth, and 34 per cent. in the blast furnace. The lead is re-melted in a kettle holding 9900 lb., poled, and drawn off into molds which rest on a car traveling on a track. The plant was idle in 1914.

6. *Joplin, Mo.*—The plant of the Lone Elm works are taken up in §70.

7. *Galetta, Ont.*—A new plant with the Newnam ore-hearth has been put into operation at Galetta, Ont. The plant[3] is shown in plan, elevation, and section

[1] Perkins, *Eng. Min. J.*, 1907, XXXIV, 388.
Buskett, *Min. Eng. World*, 1908, XXIX, 917.
Pulsifer, *op. cit.*, 1914, XL, 450, 501.

[2] Ruhl, *Min. Eng. World*, 1910, XXXIII, 1047.
Wittich, *Mines and Minerals*, 1911, XXXI, 709, with plan of works.

[3] Newnam, *Bull. A. I. M. E.*, April, 1917, 425.

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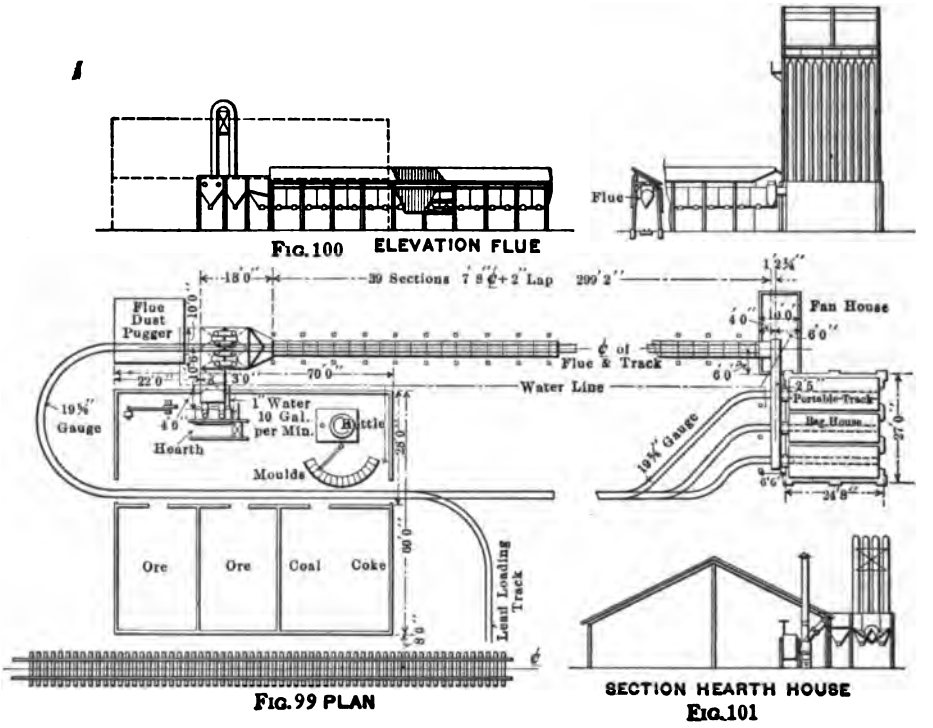
in Figs. 99–101. It contains the hearth-house with a single hearth, 8 ft. long; a drossing kettle with molding apparatus; and the necessary bins for ore, coal, and coke. The gases from the hearth pass through four goose-necks, placed in parallel, in which the temperature is greatly reduced; travel through a balloon flue 300 ft. long provided with discharge-hoppers; enter a No. 10 Sirocco exhaust fan which delivers 8000 cu. ft. per min., at a temperature of not over 87° C., to the bag house with 99 cotton bags, 18 in. in diameter and 30 ft. long. The dust from the goose-neck, hoppers, and flue-system is collected in closed buggies, pugged with lime, and bedded with the galena concentrate and the burnt bag-house dust and fume. The power required is 15 h.p.



Figs. 99 to 101.—Plan, elevation and section, Newnam Ore-hearth Plant, Galetta, Ont.

The ore treated is a mixture of 50 per cent. jig- and 50 per cent. table-concentrate averaging Pb 79.0, Insol. 0.6, Fe 1.2, CaO 0.8, Zn 2.0, S 14.0 per cent., and Ag 1.14 oz. per ton. The ore is mixed with from 2 to 3 per cent. finely crushed limestone, and about 3 per cent. coke-breeze.

The hearth produces daily about 15 tons of lead. The lead contained in the products is, pig lead 74 per cent., gray slag 12, and dust and fume 14 per cent. The pig lead assays Pb 99.95, Cu 0.03 per cent., and Ag 5.06 oz. per ton; the gray slag, Pb 41.0, Insol. 11.2, FeO 13.0, CaO 11.6, S 1.7 per cent. The slag is stored and shipped when enough has been accumulated. The lead loss is 1.5 per cent. The hearth is worked by two men on a shift; there are 14 men in the plant.

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69. Examples of Smelting in the Ore-hearth.—In Table 32 are assembled the dimensions and workings of ore-hearths treated in the text as well as of others taken from the technical literature.

70. Products and Their Disposal.—The products of the ore-hearth are lead, gray slag, flue-dust, and fume.

1. *Lead.*—The bars of lead which are ladled from the pot next to the ore-hearth are not sufficiently pure to go to market. They are subjected to a simple refining process which consists in liquating to remove suspended dross, collecting the liquated lead in a kettle, poling it with air or steam[1] for 1.5 hr. or longer at a temperature below visible red, say below 470° C., to eliminate Cu and Sb which are brought to the surface, skimming the dross, and then molding the purified lead by means of a siphon, a pump, or formerly through a spout with cock extending from the bottom of the kettle through the brickwork.

A common form of liquating reverberatory furnace[2] has an inclined trough-shaped hearth, 3 by 5 ft., consisting of two cast-iron plates, 2 in. thick, which have 1-in. holes every 8 in. on the trough-line. Beneath the trough lies a cast-iron gutter, which conducts the liquated lead into the poling-kettle. The bars of lead, charged from the side into the furnace, leave behind the dross, which is raked out at intervals.

In recent years the softening furnace of the Parkes process (§231) has been erected at some large plants, as it is more economical and, with proper management, especially as to temperature, does the work as well as the trough.

The casting of lead is taken in detail in §253.

Missouri ore-hearth lead retains usually about 0.01 per cent. Cu. Several analyses of ore-hearth lead have been given in Table 3. References to other analyses are given in the footnote.[3]

2. *Gray Slag.*—This is an agglomerated mixture of metallic lead, lead compounds, gangue, and flux, of varying composition, which contains about 15 per cent. of the Pb of the ore. A mechanical analysis of Lone Elm slag gave Dewey[4] 21.45 per cent. metallic lead and 78.55 per cent. pulp. A few analyses are recorded in Table 33.

There are in operation two methods of recovering the lead; one is to smelt the slag for metallic lead, matte, and slag in a blast furnace; the other is to smelt it in the so-called slag-eye furnace for metallic lead, slag, and white oxidized fume, which is collected and sold as white paint.

[1] Bardill Apparatus, *Eng. Min. J.*, 1915, C, 969.

[2] Brinsmade, *Mines and Minerals*, 1901–02, XXII, 300.

[3] Middleton, *Eng. Min. J.*, 1905, LXXX, 10 (England).
Williams, "Industrial Report," Geol. Survey of Missouri, 1877, p. 63; *Tr. A. I. M. E.*, 1876–77, V, 326.
Dewey, *op. cit.*, 1889–90, XVIII, 687.
Pulsifer, *Min. Eng. World*, 1913, XXXIX, 681; 1914, XL, 504

[4] *Tr. A. I. M. E.*, 1889–90, XVIII, 685.

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TABLE 32.—EXAMPLES OF SMELTING IN THE ORE-HEARTH

Name	Date	Furnace: Front to back	Width	Depth	Sides, kind	Tuyères: No.	Tuyères: Diam.	Products: Ore in 24 hr., lb.	Lead in 24 hr., lb.	Gray slag in 24 hr., lb.	Per cent. lead direct from ore	Men in 24 hr.	Materials: Bituminous coal, lb.	Coke, lb.	Peat bu.	Charcoal, bu.	Wood	Lime, bu.	Reference
Rossie Works, N. Y..	1842	24″	24″	12″	Air-jacket	1	2″	9,988	7,389	.....	73.88	4	.....	.....	•	.....	¾	.....	(a)
Keld Head Mining Co., England.....	1870	23″	21″	12″	Cast-iron	1	.....	10,752	8,064	602	74.44	6	374	.....	12	.....	.....	.....	(b)
Leadhill & Warlock, Scotland.....	1895	22″	30″	6″	Cast-iron	1	3″	.....	6,720	.....	.....	6	500	.....	.....	.....	.....	.....	(c)
Weardale, England..	1903	21″	27″	8–12″	Cast-iron	1	.....	18,000	8,700	.....	61.00	.....	.....	.....	•	.....	.....	.....	(d)
Wanlockhead, Scotland.	1912	22″	30″	6″	Cast-iron	1	.....	.....	2,240	.....	60.00	6	560	.....	.....	.....	.....	.....	(e)
Granby, Mo.....	1877	22½″	24″	11″	Water-jacket	3	.....	9,000	7,500	.....	83.90	6	.....	.....	.....	27.6	.....	2.5	(f)
Hopewell, Mo.....	1877	.....	20″	12″	Water-jacket	1	.....	6,000	1,464	.....	73.20	6	.....	.....	.....	20.0?	.....	.....	(g)
Lone Elm, Mo.....	1889	22″	48″	10″	Water- and air-jacket	7	1″	27,000	12,032	5,247	45.00	6	2,160	.....	.....	.....	.....	5.40	(h)
Granby, Mo.....	1908	15″	4′	10″	Water-jacket	4	.....	14,000°	.....	2,000	75.00	4	•	.....	.....	.....	.....	•	(i)
Galena, Kans.....	1909	.....	5′	.....	Air-jacket	10	.....	21,000	.....	.....	50.00	6	•	.....	.....	.....	.....	.....	(j)
Granby, Mo.....	1911	.....	5′	.....	Air-jacket	....	.....	.....	.....	.....	65.00	.....	•	.....	.....	.....	.....	.....	(k)
Webb City, Mo.....	1911	18″	5′	7″	Air-jacket	10	.....	14,000°	.....	.....	65.00	6	•	.....	.....	.....	.....	6	(l)
Federal, Ill.....	1914	18⅞″	4′	10″	Water-jacket	8	1″	22,500	7,200	.....	60.00	6	.....	1,800	.....	.....	.....	.....	(m)
Collinsville, Ill.....	1914	18⅞″	4′	10″	Water-jacket	8	1″	21,000	6,200	.....	55.00	6	.....	1,680	.....	.....	.....	.....	(n)
Collinsville, Ill.°°...	1915	19.5″	8′	10″	Water-jacket	12	1⅝″	13,179	6,443	3,318	67.40	6	.....	474	.....	.....	.....	.....	(o)
Joplin, Mo.....	1916	20″	5′ 5″	.....	Air-jacket	....	.....	28,000	12,600	4,200	60.00	.....	1,200	.....	.....	.....	.....	55	(p)
Galetta, Ont.°°....	1917	19.5″	8′	10″	Water-jacket	12	1⅝″	40,000	30,000	.....	74.00	4	.....	1,000	.....	.....	.....	1,000	(q)

• = used. ° = works 12 hr. only. °° = Newnam mechanical ore-hearth.

(a) *Am. J. Sc. & Arts*, 1842, XLII, 169; (b) Percy, "Lead," 278–283; (c) Sexton, *Eng. Min. J.*, 1895, LIX, 175; (d) Middleton, *Eng. Min. J.*, 1905, LXXX, 10; (e) Correspondent, *Can. Min. J.*, 1912, XXXIII, 678; (f) Williams, "Industrial Report," p. 63; (g) *Ibid.*, p. 65; (h) Dewey, *Tr. A. I. M. E.*, 1889–90, XVIII, p. 674; (i) Perkins, *Eng. Min. J.*, 1907, XXXIV, 388; Buskett, *Min. Eng. World*, 1908, XXIX, 917; (j) Brinsmade, *op. cit.*, 1909, XXXI, 1029; Pulsifer, *op cit.*, 1914, XL, 450, 501; (k) Brown, *Tr. A. I. M. E.*, 1911, XLII, 402; Pulsifer, *Min. Eng. World*, 1914, XL, 450, 501; (l) Ruhl, *Min. Eng. World*, 1910, XXXIII, 1047; Wittich, *Mines & Minerals*, 1911, XXXI, 709; (m) Pulsifer, *Min. Eng. World*, 1913, XXXIX, 375; 1914, XL, 450, 501; (n) Pulsifer, *op cit.*, 1913, XXXIX, 681; 1914, XL, 450, 501; (o) Newnam, *Tr. A. I. M. E.*, 1916, LIV, 485; (p) Brinsmade, *Min. Eng. World*, 1909, XXXI, 1029; (q) Pulsifer, *op. cit.*, 1914, XL, 450, 501.

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TABLE 33.—ANALYSES OF GRAY SLAG

Smeltery	SiO_2	FeS_2	ZnS	PbS	$PbSO_4$	PbO	ZnO	Fe_2O_3	Al_2O_3	CaO	MgO	Pb	Cu	S	Fe	Ref.
Lead Hills and Warlock, Scotland	26.00	....	....	5.63	10.36	34.88	0.95	18.	10	10.00	....	....	...	...	...	1
Joplin, Mo.	12.70	0.67	3.64	14.73	5.18	33.55	14.53	2.90	0.78	11.49	0.12	....	...	...	...	2
Collinsville, Ill.	12.5	....	....	.....	.....	.....	.....	.....	....	9.	0	46.0	0.5	3.5	10	3

1. Sexton, *Eng. Min. J.*, 1895, LIX, 175.
2. Dewey, *Tr. A. I. M. E.*, 1888–90, XVIII, 685.
3. Pulsifer, *Min. Eng. World*, 1913, XXIX, 681.

Smelting for metallic lead alone[1] was formerly carried on in a small low brick blast-furnace (26 by 36 in. and 46 in. high) with a single 3-in. tuyère at the back, an external crucible, and a hood to carry off fumes into the open. The crucible was a cast-iron pot divided into two unequal parts by a partition descending nearly to the bottom. The melted charge (black slag) flowed over the larger division, filled with charcoal, into a water-tank; the lead filtered through the charcoal, collected at the bottom, and was removed at intervals from the smaller division. As the brickwork was quickly corroded, a furnace ran only a short time. From 15 to 18 tons slag, assaying 35–40 per cent. Pb, were smelted in 16 hr. giving 7,500 lb. lead, which was not soft, and waste slag; the fuel consumption was 2 tons coke and 22 bu. charcoal.

This mode of operating, being wasteful and expensive, has been replaced by normal blast furnace work. Thus at Collinsville, Ill.,[2] there are three water-jacket blast furnaces, 36 by 126 in. at tuyères, in which are treated in 24 hr. from 120 to 140 tons of charge made up of gray slag, ignited bag-house fume, roasted matte, drosses and refining skimmings from the Parkes desilverizing plant, flue-dust briquettes, and fluxes. The slag made analyses SiO_2 29.0, FeO 37.5, CaO 19.0 per cent. The gases from the blast furnaces go through the bag house.

At Federal, Ill.[3] there are also three water-jacket blast-furnaces, 40 by 96 in. at tuyères. The charges are mixtures of gray slag, ignited bag-house fume, blast roasted ore, roasted matte, and pyrite; they contain 40 per cent. Pb, and give a matte-fall of 17 per cent. The slag is calculated to contain SiO_2 32, FeO 30–32, Ca(Mg)O 16–19, ZnO 8, Al_2O_3 up to 9 per cent.; it assays 0.5–1.3 per cent. Pb.

Other works which used to smelt their gray slag in small circular blast-furnaces, find it more profitable to sell it to blast furnace plants.

Smelting of gray slag with other charge-components for metallic lead, oxidized lead fume, and waste slag is considered in §72.

3. *Flue-dust and Fume.*—The gases drawn from the ore-hearth by a powerful suction fan carry dust and fume; the larger part of the former settles in the chamber and flue leading to the fan, the rest is filtered with the fume in the

[1] Bergen, *Commissioner Raymond's Report*, 1875, 424.
[2] Pulsifer, *Min. Eng. World*, 1913, XXXIX, 681.
[3] Pulsifer, *Min. Eng. World*, 1913, XXXIX, 375.

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bag house and collected in the brick bays of the building. The dust consists mainly of fine particles of charge which may or may not have been oxidized, the fume almost entirely of oxide lead compounds. The chemical analyses given in Table 34 show the characteristics of the two products.

TABLE 34.—ANALYSES OF ORE-HEARTH DUST AND FUME

Smelter	SiO_2	PbS	ZnS	$PbSO_4$	PbO	ZnO	Fe_2O_3	Al_2O_3	CaO	MgO	CO_2	SO_2	S	Pb	C	Ref.
Lone Elm, Mo.																
Chamber-dust	2.76	51.59	5.57	22.74	3.79	0.35	0.93	0.06	6.40	0.03	3.83	0.86	....	.....	...	..
Flue-dust	2.54	10.41	0.11	61.39	11.47	0.42	0.97	0.05	5.24	0.03	1.35	none	....	.....	...	..
Dust from cooling pipe	2.26	.....	2.68	89.84	1.20	....	1.94	0.32	0.27	....	0.13	0.18	....	.....	...	..
Raw or blue fume	....	.....	....	.....	.....	....	....	....	....	....	....	.....	....	.....	...	..
Row of bags 1, hopper 1	0.17	6.18	Zn 0.61	45.34	44.44	....	0.10	0.10	0.21	....	0.23	0.96	....	.....	...	1
Row of bags 2, hopper 6	0.11	10.37	0.34	43.55	44.48	....	0.05	0.05	0.01	....	0.19	0.13	....	.....	...	..
Row of bags 4, hopper 1	0.12	5.19	0.43	46.88	45.08	....	0.07	0.07	0.03	....	0.26	0.68	....	.....	...	..
Row of bags 4, hopper 10	0.14	8.61	0.61	43.57	44.18	....	0.07	0.07	0.02	H.O	0.08	0.44	....	.....	...	..
Sintered fume	0.10	.....	....	48.76	46.82	0.27	0.32	0.05	0.48	0.37	0.90	1.65	....	.....	...	..
Collinsville, Ill.						Fe										
Raw fume	5.0	.....	....	.....	.....	4.6	....	....	2.5	2.5	....	.....	6	75	...	2
Joplin, Mo.																
Raw or blue fume	0.24	4.63	....	47.68	42.24	0.75	0.52	0.52	....	....	....	0.496	....	.....	3.4	3
Burnt blue fume	....	.....	....	.....	.....	2.90	....	....	....	....	....	.....	5.71	74.90	...	3

1. Dewey, *Tr. A. I. M. E.*, 1889–90, XVIII, 674.
2. Pulsifer, *Min. Eng. World*, 1913, XXXIX, 681.
3. Buskett, *Min. Eng. World*, 1912, XXXVI, 393.

The chamber and flue-dust have to be agglomerated before they can be successfully smelted. Some works resort to briquetting with slaked lime (§216), others to sintering, others add the dust to blast roasting charges; moistening the dust and adding it to the blast furnace charge is an inefficient method of handling.

The fume was formerly allowed to go to waste. In 1876 the Lone Elm works of Joplin, Mo., began to filter and convert it into a white paint (Sublimed Lead) by sintering and then smelting with gray slag in a slag-eye furnace for lead, slag, and fume which was collected in a bag house.

The plan of the former Lone Elm works[1] is shown in Fig. 102. The gases from the ore-hearth are drawn off by a suction fan, 6 ft. in diameter and 3 in. wide, making 290 r.p.m.; pass through a water-jacketed brick flue into a brick dust-chamber (40 ft. long, 19 ft. high, 6.5 ft. wide, with a door on one side) which collects coarse particles of more or less changed ore and fuel; leave the chamber at the top and travel through a horizontal sheet-iron pipe (350 ft. long, 5 ft. in diam., supported by 20-ft. pillars) to the fan; and thence through a 4-ft. pipe, resting on 12-ft. pillars, to the blue-powder or blue-fume bag house. This first bag house is similar in construction to the second, the white-paint bag house, shown in

[1] Dewey, *Tr. A. I. M. E.*, 1889–90, XVIII, 674.

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cross-section in Fig. 103. It is a brick building (95 ft. long, 50 ft. wide, and 45 ft. high) divided into two compartments by a longitudinal wall, so that one may be shut off when it is necessary to gain access to the bags. Each compartment is divided into two stories, the lower being 12 ft. high. The divisions (columns, beams, etc.) are all made of iron pipe. In fact everything in the

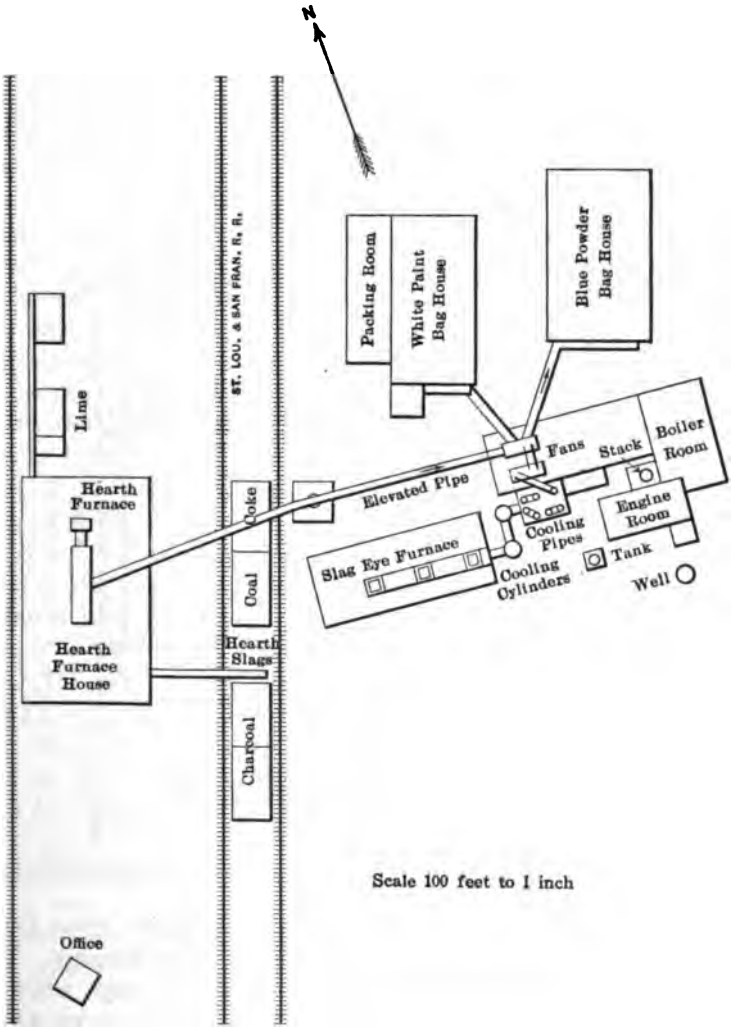

FIG. 102.—Plan of Lone Elm Works, Joplin, Mo.

building is either of brick or iron, except the filtering bags. The lower story contains four rows of sheet-iron hoppers, extending the length of the building, which serve to collect the fume that has accumulated. They have the form of a truncated pyramid and are closed at their lower faces by sliding dampers. They stand on four pipes, 3.5 ft. long, encased in refractory clay pipes. The upper face of a hopper is covered with sheet-iron $\frac{1}{16}$ in. thick. This has 16

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holes, 18 in. in diameter, from which thimbles, 12 in. high, project upward. Over these the lower ends of the bags, made of unwashed wool, 60 in. in circumference and 33 ft. long (changing to 50 in. and 35 ft. when in use), are slipped and tied fast. The upper ends are tied with strong cord, with which they are suspended from beams near the roof. There are 800 bags in the bag house. Between every two rows of bags is an iron scaffolding with iron footways placed at convenient heights, so as to make all parts of the building accessible.

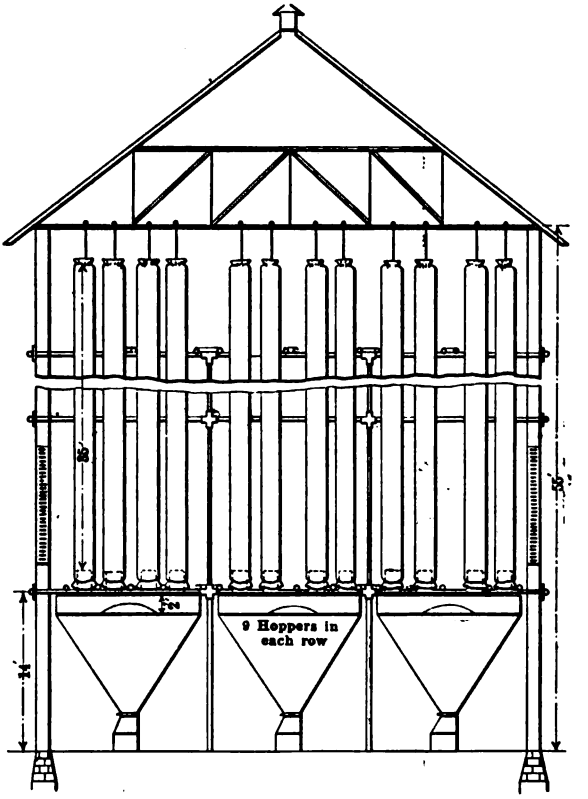

FIG. 103.—Section of bag-house for white fume, Lone Elm Works, Joplin, Mo.

The cooled gases, being forced through the main pipe, enter four branch pipes, each of which passes through and connects a set of hoppers. The gases, laden with fume and some dust, ascend into the hanging bags, where they are filtered, the fumes falling into the hoppers below. These are emptied once in 2 days, when the bags are also shaken to detach adhering fume. For this purpose the current of the gas is shut off, and men with aspirators pass quickly through the building, giving each bag a quick shake.

The collected fume is a very fine bluish-gray powder (blue powder), consisting mainly of PbO and $PbSO_4$, with some PbS.

The fume is sometimes sold as Sublimed Blue Lead to be used in the manufacture of rubber (black tread) tires, or as a paint. Usually, however, it is let

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down from the hoppers, spread over the floor in piles, and set on fire with oil-waste. Often it fires itself. It burns (smolders) for about 10 hr. but does not flame; a good deal of heat is liberated, and some SO_2 set free. The fine, loose blue powder is converted into a porous, pinkish-white crust that is still friable but sufficiently coherent to stand handling and charging. The roasted blue powder is free from carbonaceous matter and PbS. Analyses are given in Table 34.

In refining the roasted blue powder in the slag-eye furnace, the object is to oxidize all the components of the charge as much as possible; hence little metallic lead is produced. To prevent any carbonaceous compounds from injuring the color, hard 72-hr. coke is used as fuel. This subject is discussed in §72.

71. Cost.—The cost of smelting 65- to 70-per cent. galena concentrate in the ore-hearth, collecting fume in bag house, and smelting gray slag and sintered fume in the blast-furnace for pig lead, matte, and slag is about $6.00 a ton.[1] The yield in lead is about 98 per cent. Brown[2] gives the cost at Granby, Mo., for smelting ore and sintered fume in the ore-hearth (the gray slag being sold) as $3.30 per ton of lead-bearing material.

72. Lewis and Bartlett or Sublimed-Lead Process.[3]—This process, which was started at Joplin, Mo., in 1876, is in operation in the United States at two plants, that of the Eagle-Picher Lead Co., Joplin, Mo., and that of the St. Louis Smelting & Refining Co., Alton, Ill.; and in England at the works of the Bristol Sublimed Lead Co.[4]

The aim of the process is to volatilize the lead from galena and the gray slag and burnt blue powder of the ore-hearth, by smelting in the oxidizing atmosphere of a slag-eye furnace, and to recover the oxidized fume in a bag house. The products are waste slag, some metallic lead, and so-called sublimed lead, a finely divided white powder with about 80 per cent. $PbSO_4$ and 20 per cent. PbO, which is used for paint, the manufacture of oil-cloth, stained paper, and rubber goods.

The two American smelteries are very similar in their general arrangements. A plan and elevation of plant are shown in Figs. 104 and 105.[5] The brick building, 215 ft. long and 75 ft. wide, has four divisions: (1) The furnace-room, 75 ft. long, with two slag-eye furnaces, *A*, and a single combustion-chamber above; one large and two small cooling-towers, *B* and *B'*; two No. 7 Sturtevant blowers, *H*, driven by two 30-h.p. A.C. motors to furnish blast at

[1] Ingalls, *Tr. A. I. M. E.*, 1906, XXXVII, 629.
Finlay, *Eng. Min. J.*, 1908, LXXXVI, 607.
[2] *Tr. A. I. M. E.*, 1911, XLII, 402.
[3] Dewey, *Tr. A. I. M. E.*, 1889–90, XVIII, 674.
Blair, *Eng. Min. J.*, 1910, XC, 906.
Hughes, *J. Soc. Chem. Ind.*, 1909, XXVIII, 415.
Buskett, *Min. Eng. World*, 1911, XXXV, 57; 1912, XXXVI, 393.
Schaeffer, *J. Ind. Eng. Chem.*, 1913, V, 149.
[4] Roesing, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1888, XXXVI, 103.
[5] Blair, *Eng. Min. J.*, 1910, XC, 906.

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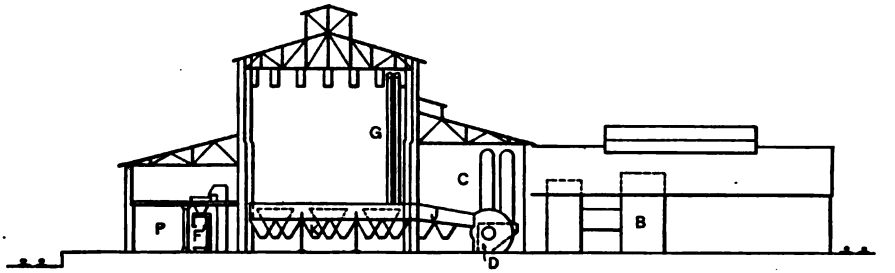
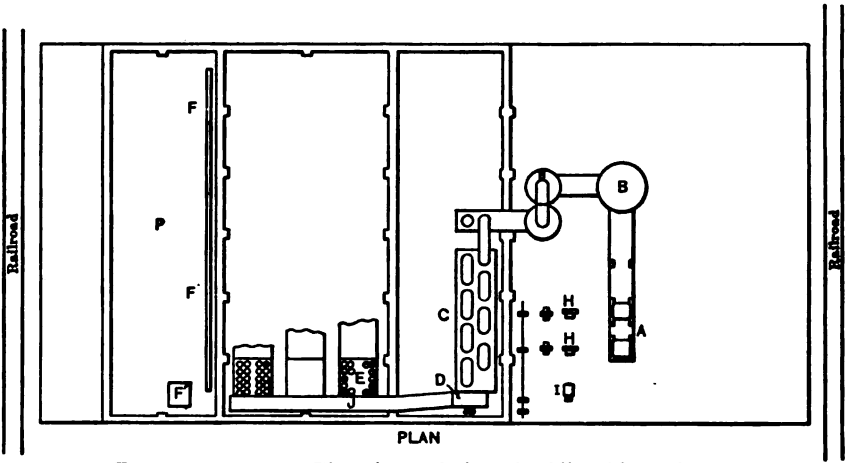
4½ oz. for the slag-eye furnaces; and one freight elevator, *I*, driven by a 20-h.p. A.C. motor.

(2) The goose-neck room, 30 ft. long, with seven goose-necks, *C*; one 7-ft. fan, *D*, driven by a 50-h.p. motor, delivery-pipe, and storage space for clay and other materials.

(3) The bag room, *G*, 75 ft. long, with delivery-pipe, *J*, coming from goose-neck room with branch-pipes leading into three rows of hoppers, *K*, nine hoppers in a row, each with 20 filter-bags, *E*.



ELEVATION



PLAN

FIGS. 104 and 105.—Elevation and plan of sublimed-lead plant.

(4) The packing-room, *P*, 30 ft. long, with screw-conveyor, *F*, bucket-elevator and packer, *F'* (driven by a 20-h.p. motor); and cooper-shop.

The slag-eye furnace of the Lone Elm works[1] first erected was built of brick with water-boxes, and had two rows of tuyères; it has been replaced by an oval water-jacket furnace with a single row of tuyères. A plan and elevation of the jackets are shown in Figs. 106 and 107, and an elevation of a pair of furnaces with combustion chamber in Fig. 108.

[1] Dewey, *loc. cit.*
Hofman, "Lead," preceding edition, 128–130.

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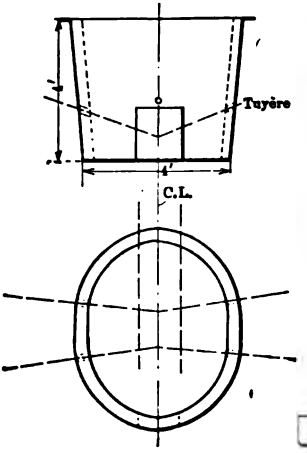
The oval water-jacket (Figs. 106 and 107) is 4 ft. high, 5 by 4 ft. at the throat and 4.5 by 3.5 ft. at the base. At the front it has a tapping-jacket, 18 by 18 in.; at 18 in. from the bottom are two 2.5-in. tuyères on a side, which are 18 in. apart and point downward, and two 2.5-in. tuyères at the back which are 12 in. apart. The jacket (Fig. 108) rests on a brick base which extends 6 in. upward inside the jacket. The smelting charges are introduced through the feed-door, *B*, 1 ft. high and 2 ft. wide, which serves also as port for the admission of the air necessary for the oxidation of the fumes arising from the hot top of the charge. Slag and lead run continuously from the tapping-jacket into the fore-hearth, *C*, which collects the lead, while the overflowing slag, with 2 per cent.



FIGS. 106 and 107.—Elevation and plan of water-jackets.

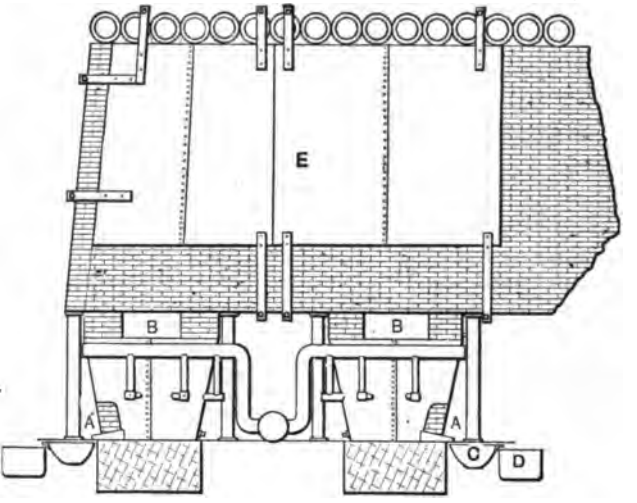


FIG. 108.—Elevation of a pair of slag-eye furnaces with combustion chamber.

Pb, is granulated by an impinging stream of water and carried off in the trough, *D*. The lead is ladled out at intervals. The lining of a furnace lasts about 3 months. Above the pair of furnaces is the braced combustion-chamber, *E*, supported by iron columns and covered with fire-clay pipes, 4 ft. long and 10 in. in diameter, laid in fire-clay. Being open at the ends, the pipes are kept cool by the circulating air. At one plant the space between the furnace and the combustion-chamber is water-jacketed instead of being built of brick as in the figure.

The first cooling and condensing brick tower, *B*, Figs. 104 and 105, is 20 ft. high and 10 ft. in diameter. It is lined with fire-brick, braced with seven 60-lb. rails, and tied by three 1.5-in. iron rods. The inside temperature is held at 800° C. The two smaller towers, *B'*, are soft-steel shells lined with 4-in. red brick, and connected at the top by an iron pipe, 44 in. in diameter. Each tower has a cleaning door near the bottom. The gases from the combustion-chamber enter tower, *B*, near the top, pass off near the base into a short flue leading into one of the towers, *B'*, in which they ascend, and after traveling through the connecting pipe descend in the other. They then pass into a flue with two

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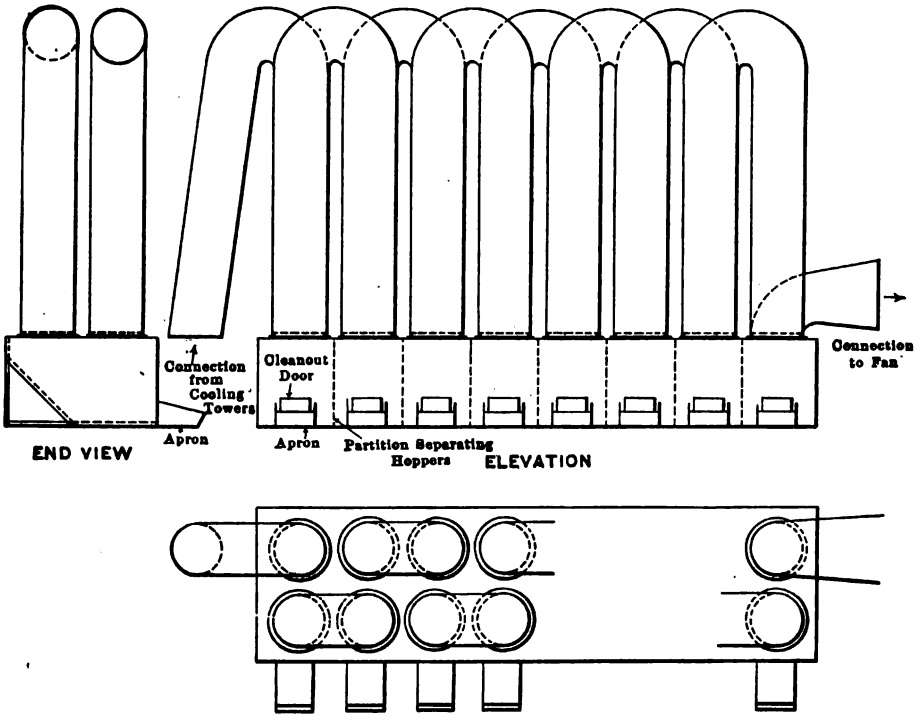
openings in the roof, one of which is connected with a stack used temporarily when the gases are to escape into the open, while the other receives one limb of the connecting goose-neck, *C*.

There are seven goose-necks, made of sheet-iron, gauge No. 14, which are to cool the gases and to collect the dust not deposited in the towers. A plan, side- and end-elevation of a set of goose-necks are shown in Figs. 109–111. A goose-neck is 44 in. in diameter and 25 ft. high, and rests on a casing having at the front cleaning doors and apron-plates for the removal of settled fume.



FIGS. 109 to 111.—Plan, side- and end-elevations of a set of goose-necks.

This has a pinkish color. It is collected daily to the amount of 3500 lb. and is either smelted or sold as an inferior pigment, mainly to the rubber trade.

The gases, which have been cooled and freed from dust and some fume, pass through the fan, *D*, Figs. 104 and 105, into the delivery-pipe, *J*, and from this through three branches into three rows of hoppers, *K*, in the bag house. Each hopper has, Fig. 103, 20 bags, 20 in. in diameter and 30 ft. long, made of unwashed wool, making a total of 540 bags. They are shaken twice in an 8-hr. shift. The pigment which collects in the hoppers is either discharged into the screw-conveyor, *F*, which delivers to the packer, *F'*, or it is carted into the packing-room and shoveled into barrels which hold 500 lb. paint. The advantage of the conveyor is that by working, it frees the paint from mechanically included

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gases and thus makes mechanical packing possible; further, the work is less unsanitary than if done by hand. A similar arrangement is discussed in §73.

In blowing-in a slag-eye furnace, this is warmed with wood, then filled with coke, and the blast turned on. When the coke has become red-hot on the surface, the first ore-charge is given. This is fed in small shovel-lots at a time so as not to cool the red-hot top; feeding is therefore a more or less continuous operation. The components of the charge and the manner of making up the charge vary somewhat. Galena alone can be smelted with coke, but furnace-troubles are likely to occur, and off-color pigment may be formed. In smelting sintered blue fume alone an almost pure basic sulphate is produced. The usual three important Pb- and S-bearing components are raw galena, gray slag, and burnt fume; to these may be added lead-bearing materials, such as drosses, skimmings and scrapings, off-color paint, carbonate ore, etc. Charges smelted in 24 hr. consist[1] of carbonate ore, 1000 lb.; ashes from cooling towers, 1500; burnt blue fumes from ore-hearth, 2000; galena concentrate free from blende, 6000; off-color pigment, 1000; cuttings (*i.e.*, accretions) from combustion-chamber, 1000; gray slag from ore-hearth, 1000; metallic iron (tin cans, sweepings from machine shop, puddle cinder) 900; limestone (crushed through 40-mesh screen), 1400; total charge, 15,800 lb.; coke, 6000 lb. Blair[2] gives the following: Ashes, 2000 lb.; burnt blue fume, 3000; galena, 6000–7000; off-color pigment, 1000–1500; cuttings, 1000; gray slag, 2000; iron, 900; limestone, 1400; black slag (from slag-eye furnace, too rich in Pb to go to the dump), 1000–3000 lb. It will be noted that carbonate ore is replaced by black slag. In general, the charges are made up with the view of obtaining an abundance of fume of correct composition, and a slag that will run freely and carry little Pb.

The feeding of the charge-components is sometimes given into the hands of the feeder who uses his own judgment as to requirements. Buskett[3] recommends to weigh out the charge components, and mix them, with the exception of galena and coke. A single charge would consist of galena, 500 lb.; gray slag and other oxide lead material, 450; silica, 10; puddle cinder, 130; limestone, 10; total charge, 1100 lb.; and coke 135 lb.

In feeding, the attendant first introduces the coke and allows this to burn freely; scatters the galena over the top in a thin layer and sees that this burns satisfactorily; he then covers it with mixture; and finishes with the bed of coke of the next charge.

Beside the make-up of the charge and the manner of feeding there have to be considered the temperature of the top of the charge and of the combustion-chamber, and with this the volume of air blown in through the tuyères and drawn in through the feed-door, if a pigment of the right composition and color is to be obtained. Beside having a hot top and a liquid slag, it is important that no irregularities or stoppages occur in furnacing, as otherwise the paint obtained

[1] Buskett, *loc. cit.*
[2] *Loc. cit.*
[3] *Loc. cit.*

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will be of an inferior grade. Thus, when a furnace has been started, the product of the first 4 or 5 hr. is classed as blue powder, and that of the next 20 hr. is still off-color. The main products of the furnace are slag (black slag), sublimed lead, and metallic lead. Table 35 gives analyses of black slag.

TABLE 35.—ANALYSES OF BLACK SLAG FROM SLAG-EYE FURNACE

Plant	SiO_2	FeO	CaO	ZnO	Pb	Reference
Eagle-Picher Lead Co..	27.8	40.7	20.4	.....	2.1	Buskett, *Min. Eng. World*, 1912, XXXVI, 398.
Eagle-Picher Lead Co..	26.2	42.5	18.6	.....	2.6	
Eagle-Picher Lead Co..	32.2	36.0	12.5	.....	.....	W. W. Petraeus, private com., 1895.
Eagle-Picher Lead Co..	27.5	33.5	22.0	8.05	4.0	
St. Louis S. & R. Co..	25.0	32.5	17.5	.....	.....	H. Garlichs, private com., 1914.

The sublimed lead is essentially a basic lead sulphate with 75–90 per cent. $PbSO_4$, and 25–10 per cent. PbO; there are usually present 2–5 per cent. ZnO; the content of $ZnSO_4$ ought not to exceed 1 per cent.; and that of CO_2 0.07 per cent. Analyses are given in Table 36.[1]

TABLE 36.—ANALYSES OF SUBLIMED LEAD PIGMENT

$PbSO_4$	81.4	78.5	82.6	79.6	84.0	80.9	88.3	73.5	77.9
PbO	14.9	17.0	13.4	15.0	10.1	14.2	8.5	20.5	16.5
$ZnSO_4$	1.0	1.0	1.0	2.8	2.1	0.6	0.32	1.8	2.6
ZnO	2.3	3.4	2.9	2.5	3.5	3.8	2.2	4.1	2.8
SO_3	0.082	....	....	.....	....	0.032	0.06	None	None
CO_2	trace	....	....	.....	....	.....	.....	None	None
H_2O	0.2	....	....	0.1	0.2	.....	.....	0.1	0.2
	99.882	99.9	99.8	100.0	99.9	99.532	99.38	100.0	100.0

In order to obtain a basic sulphate of correct composition it is essential that the temperatures be correct. It has been shown in §40 that $PbSO_4$ heated in a current of air is in part dissociated at 705° C. and that decomposition increases with temperature, hence the importance of having a high temperature in the combustion-chamber and of holding the first cooling tower at about 800° C.

The lead produced by the slag-eye furnace is hard and whitish on account of the impurities taken up in the smelting.

The cost of a plant in the Mississippi Valley, treating 40 tons ore per day of the character shown in the drawings, is estimated by Buskett[2] to be about $100,000 distributed as shown in Table 37.

[1] Buskett, *loc. cit.*

[2] *Loc. cit.*

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TABLE 37.—COST OF A 40-TON SUBLIMED LEAD PLANT

Railways and grading	$8,000
Office and laboratory	4,000
Crushing plant	4,000
Furnaces and building	20,000
Cooling system	7,000
Goose-neck house	3,000
Packing house	5,000
Bag house	24,000
Storehouse	5,000
Power plant	20,000
	$100,000

The daily operating expense is given in Table 38.

TABLE 38.—COST OF OPERATING A 40-TON SUBLIMED-LEAD PLANT

Labor:		
1 manager		$10.00
1 superintendent		7.00
1 cashier		3.33
1 chemist		3.33
1 assistant chemist		1.50
1 porter		1.50
3 foremen		9.99
6 feeders		12.00
6 tappers		12.00
6 tappers' helpers		12.00
6 pot pullers		12.00
3 mixers		6.00
12 wheelers		19.20
5 yardmen		8.00
3 bag-room men		6.00
1 packer		2.50
Cost of labor, total		$134.55
Cost of labor, per ton of ore	$3.36	
Materials:		
40 tons ore at $60.00		$2,400.00
Fluxes		12.50
12 tons coke at $2.50		30.00
70 packing barrels at 0.35		24.50
Cost of materials, total		$2,467.00
Cost of materials, per ton of ore	$61.67	
Power		24.00
Total expense per day		$2,625.55
Production and Profit		
35,500 lb. pig lead at 4.5 cts		$1,597.50
35,000 lb. sublimed lead at 5.5 cts		1,925.00
Total production		$3,522.50
Total expense		$2,625.55
Profit		$896.95

These figures do not include sales cost, interest on investment, amortization, and repairs.

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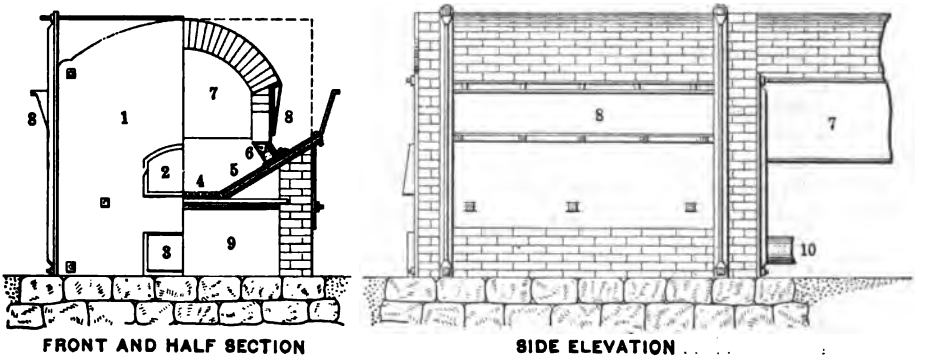
73. F. L. Bartlett Process for the Treatment of Complex Zinc Lead Sulphides.[1]—The process, formerly carried out at Cañon City, Colo., started in 1896, and before that date at Portland, Me. (burnt in 1895), aims to oxidize and volatilize the zinc and lead of an ore, the resulting fumes to be saved and then refined to a pigment; to collect the precious metals in a Cu-Fe matte; and to dispose of the gangue in the form of a waste slag.

The Ore.—The ores commonly treated are an intimate mixture of silver- and gold-bearing sulphides of zinc, lead, and iron, in varying proportions, with about 20 per cent. gangue. They are divided into two classes: those containing under 20 per cent. Zn to be smelted raw in the blast-furnace and those containing over 20 per cent. Zn to be worked in the sintering or blowing-up furnace.

Sintering of Ore and Volatilization of Lead and Zinc.—The raw ore, crushed fine and mixed with bituminous coal, is charged on a perforated grate forming the bottom of an arched chamber red hot from a previous charge; under- and



FIGS. 112 and 113.—Bartlett blowing-up furnace for zincky ores.

over-grate blasts are then let on, which start the oxidation and raise the temperature, and thereby change the loose sulphide ore into a more or less oxidized and sintered mass; the fumes pass off through a flue and are cooled and collected.

The furnace, called a "blowing-up furnace," is shown in Figs. 112 and 113. It consists of a perforated grate, 6 ft. long and 3 ft. 6 in. wide (4 and 5), resting on the side walls of the furnace and supported by cross-bars. It divides the arched chamber into the ash-pit, closed by the ash-pit door (3), and the

[1] While the process does not strictly belong to a treatise on lead metallurgy, as it extracts the precious metals rather by matting than by lead smelting, it has enough points in common with the hearth treatment of lead ores, with the collection of silver-bearing fumes by filtering, and with other points of lead smelting, to more than justify its insertion in this book.

Bartlett, *Colo. School Mines Sc. Quart.*, 1892–93, II, No. 1, 1; *Min. Ind.*, 1896, V, 619; *Eng. Min. J.*, 1889, XLVIII, 94; 1893, LVI, 3, 336; 1896, LXI, 492, 587, LXII, 172.

Hofman, *op. cit.*, 1893, LVI, 447.

Hawker, *op. cit.*, 1893, LVI, 94.

Longmaid and Collins, *op. cit.*, 1896, LXI, 587, LXII, 243; Private notes, 1896.

Correspondent, *Min. Eng. World*, 1908, XXIX, 167 (Works U. S. Smelting Co.).

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hearth with its working door (2) and flue (7). The roof is supported by channel irons (8), and its brick side walls (later replaced by water-jackets) rest on hollow columns (6), which are perforated on the hearth side to admit air under pressure on the charge. The ore mixture fills the charge-pockets (8) and passes between the columns (6) on to the hearth. The blast, entering at (10), is admitted under the charge through (4), at the side of it through (5), and on top of it through (6), thus insuring the desired degree of oxidation of the charge and especially of the fumes.

The ore containing over 20 per cent. Zn is crushed through a 2-mesh sieve, mixed with from 15 to 20 per cent. slack coal, and charged into the pockets. Supposing a previous charge to have just been drawn, the furnace to be red hot, and the charge-pockets filled, the furnace-man will push the charge down on the grate, spread it to a depth of about 6 in. through the working door (2), start the blast; and fill again the pockets. On arriving on the grate the charge will consist of ore and coke, as the coal has given up its volatile hydrocarbons while it occupied the lower part of the pocket and the upper part of the inclined grate (5). The coke ignites, and starts the roast-smelting of the charge; the flames, at first blue from CO, soon become white from the lead—later from the zinc fumes; the temperature rises, the charge begins to clinker, and when the fumes have ceased to be evolved, about 30 min. after charging, the blast is shut off and the clinker drawn out through the working door, when the furnace is ready for the next charge. Nearly all of the lead and most of the zinc have been driven off, and the precious metals are contained in the matte, which with the slag forms the clinker. Any parts of the charge that are not agglomerated are returned to the furnace and worked with the next charge. It takes from 20 to 40 min. to treat a charge, the blast has a pressure of from 4 to 8 oz. per sq. in., a furnace puts through about 6 tons in 24 hr., and one man attends to two furnaces. The composition of the clinker, of course, must vary somewhat; it contains less than 1 per cent. Pb. Analyses in Table 39 show the other constituents:

TABLE 39.—ANALYSES OF CLINKER FROM BARTLETT PROCESS

FeO	36.4	45.2	50.5	30.5
Insol	30.2	27.5	28.0	30.6
S	10.6	7.7	8.6	5.6
C	3.2	3.5	3.2	3.0
Zn	19.5	15.5	10.2	14.7
Cu	1.0	trace	trace	2.0

Smelting of Sintered Ore and Volatilization of Zinc.—The sintered ore is smelted in a low water-jacketed blast furnace with external crucible. The furnace is worked with a hot top in order to volatilize as much as possible of the zinc which is to be cooled and collected; its action is oxidizing rather than reducing; copper ore is added to assist in collecting the precious metals in the matte and to make a good separation of it from the slag. The furnace, 108 by

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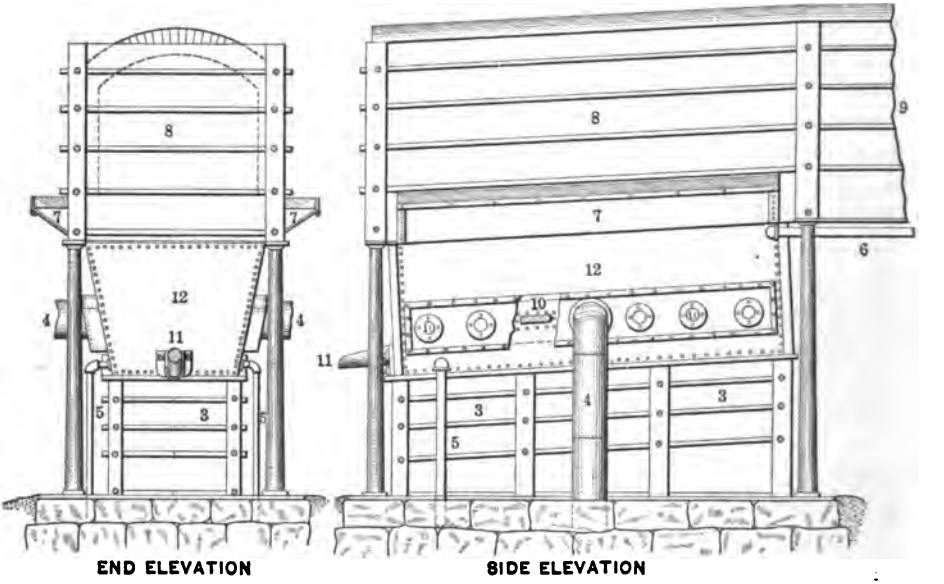
36 in. at the tuyères, is shown in Figs. 114 and 115. In the figures, (3) represents the foundation walls well bound with iron, (4) the blast-pipe ending in an air-chest attached to the water-jackets, the blast preheated by being conducted through the flues, (5) the water-inlet, and (6) the water-outlet pipes, (7) the charge-pockets, the charge sliding down the jackets, making an angle of about 45°, and being in the center only about 12 in. deep above the level of the tuyères, (8) the shaft held together by corner irons and tie-rods, (9) the flue collecting the fumes from two furnaces and leading them to the cooling chambers and bag rooms, (10) the tuyère openings, longitudinal slots, 8 by 1¼ in., the admission of air being regulated by a block pushed in or drawn out by a small iron rod attached to it, (11) the spout over which the melted masses flow into an overflow slag-pot, the bottom of the furnace slanting 1 in. to the foot from back to front, (12) the four steel water-jackets, 40 in. high.



FIGS. 114 and 115.—Bartlett blast furnace for zinc ores.

The furnace charge is made up by the addition of sulphide ores of zinc, lead, and copper, and of some silicious ores in proportion to contain 17 to 20 per cent. Zn, 2.5 to 4 per cent. Cu, 3 to 10 per cent. Pb, 15 to 20 per cent. S, and 10 per cent. lime, the rest being SiO_2 and Fe. The amount of coke used ranges from 6 to 15 per cent., varying with the percentage of Zn that is to be driven off as fume. The furnace is run in such a way as to leave enough S in the charge to bind all the Cu and the remaining Zn and thus minimize the loss of Ag by volatilization. The blast pressure is 12 oz. per sq. in. The furnace puts through from 40 to 75 tons of charge in 24 hr., the amount depending upon the percentage of Zn present. Thus, with 20 per cent. Zn 40 tons are smelted, with 15 per cent. 50 tons, with 12 per cent. 75 tons. Three men attend the

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furnace on a shift. A furnace is blown out once a month, as it is necessary thus often to clean out the flues that have become choked with dust. The matte produced, 1 ton from 10 to 12 tons of ore, runs about 40 per cent. Cu and averages 125 oz. Ag and 2 oz. Au per ton. The slags aimed at are silicious, as shown by the analyses in Table 40.

TABLE 40.—ANALYSES OF SLAGS FROM BARTLETT PROCESS

	Straight iron slags			Iron and manganese slags						Iron and lime slags						
Insol.....	32.4	36.5	33.6	36.2	38.0	38.5	39.0	40.5	44.2	38.0	39.7	36.5	32.5	37.0	38.2	37.5
FeO.....	50.5	49.3	47.2	34.7	34.6	33.6	32.5	31.9	36.3	30.1	39.5	30.3	31.7	30.1	34.3	34.2
ZnO.....	15.2	12.5	14.2	13.0	14.2	12.5	11.8	11.2	7.2	11.4	10.1	9.3	9.3	8.2	3.7	3.1
CaO.....	....	....	2.2	4.0	2.5	4.6	4.2	4.0	2.0	15.4	7.8	18.5	18.5	19.4	20.4	19.2
MnO.....	....	....	....	7.8	8.1	7.0	8.0	8.9	7.4	3.1	....	2.0	4.2	1.3	3.0	2.1
Silver, oz..	2.5	1.75	2.0	1.5	1.0	0.75	0.3	0.25	0.25	0.25	0.25	0.5	0.75	0.25	0.25	0.25

Bartlett finds that ZnO is not present in the slag as a silicate, an opinion in which he does not stand alone. Basic iron slags dissolve as much as 20 and 25 per cent. ZnO. Though the dissolving power decreases with the acidity of the slag, a certain proportion of CaO aids in the solution of ZnO; but if it goes above 15 per cent. it hinders it. The slags are made to average from 7 to 8 per cent. ZnO, as a higher percentage drags Ag into the slag. The Ag-content ordinarily ranges from 0.25 to 1.25 oz. per ton, depending on the presence of Cu and CaO, the lower figure referring to a charge with 4 per cent. Cu, the higher to one with 1 per cent. Cu, a slag high in CaO being low in Ag.

Cooling of Gases, Settling of Dust, and Filtering of Fume.—The gases passing off in sintering the raw ore and in smelting the sintered product carry with them in the form of dust fine particles of the charge more or less oxidized, fumes of lead and zinc in the form of $PbSO_3$, $PbSO_4$, PbO and of ZnS, $ZnSO_3$ and ZnO, further SO_2, CO_2, free O and N. If they are allowed to expand and to cool, and are made to change their direction repeatedly and at the same time to strike upon large surfaces, they will readily drop the dust, but the fume can be satisfactorily collected only by filtering. It has been found that the Ag carried off by the gases is contained mainly in the dust, so that, with the dust well settled, the loss in Ag by fume will be very small. The result is that under normal conditions the loss in Ag is a fixed quantity, be the ore charge high or low in Ag; thus the fume from the sintering furnace assays 5 oz. and from the blast-furnace 8 oz. Ag per ton. These figures will increase with oxidize zinc and lead ores on account of the lack of S, and also with arsenical and antimonial ores; they will decrease with sulphide zinc, iron, and copper ores, and increase slightly with sulphide lead ores and in the presence of CaO; while, as stated above, the CaO in the slag decreases the Ag carried off by the latter. The total loss in Ag in treating a charge containing 4 per cent. Cu is ¾ oz. per ton, which may increase up to 3 oz. per ton if only 1 per cent. Cu be present.

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There is a small gain in Au; the amount of Pb collected in the fume shows an increase over that accounted for by the dry assay; the loss in Zn amounts to 5 per cent. and is accounted for by the assay of the slag.

The gases from sintering and smelting are drawn off by Sturtevant fans placed back of a brick chamber having a vertical partition extending upward from the floor. The object of the chamber is to mix the gases from the sintering and smelting furnaces, and to equalize their temperatures. This is necessary, as the fans sucking off hot fumes would create a back pressure on those taking in cold fumes. In order to cool the gas mixture the fans suck in cold air. The cooled and diluted gases of approximately even temperature and composition are forced into an iron chamber. They enter at the bottom, are deflected by the roof and drop a large amount of dust. They then pass through a pair of oblong sheet-iron cooling flues, 8 by 3 ft. and 1400 ft. long, which are supported in the air on a low roof-shaped trestle so that the dust, that settles out, about 3 per cent. of the weight of the ore, may be easily removed through doors at the sides and raked into wheelbarrows or trucks. Generally 25 sq. ft. of sheet iron cooling surface are required for 1 sq. ft. of grate area of sintering furnace. At the ends of the cooling flues the gases, freed from dust, pass into one of the two bag houses used alternately, where the fumes are filtered.

A bag house, similar in construction to the one shown in Fig. 103, contains 1500 bags made of loosely woven cotton or wool. The bags are 20 in. in diameter and 21 ft. long. Cotton bags last from 18 to 24 months, and woolen bags last from 6 to 10 years. With cotton the temperature of the bag room should not exceed 90° C., and with wool 120° C. Generally 200 sq. ft. of cloth are required per square foot of grate area of sinter furnace, and if twice cleaned or shaken in 24 hr., 1 sq. yd. of woolen cloth will collect 1 lb. of fume per day, 1 sq. yd. of cotton cloth $\frac{2}{3}$ lb.

The raw fume averages:

$$12PbS,\ 30PbSO_4(+PbSO_3[1]) = 42 \text{ per cent.}$$
$$14ZnO + 40ZnSO_3 = 54 \text{ per cent.}$$
$$1C,\ 2SO_2,\ 1SiO_2, \text{ etc.} = 4 \text{ per cent.}$$

Refining Raw Fume into Refined Zinc-lead Pigment.—The object of the refining process, which is an oxidizing muffle-roast, is to convert the lead and zinc compounds into $PbSO_4$ and ZnO, to eliminate finely divided C, As, Cd, and other impurities, and to compact the fume by grinding that it may be well adapted for use as a pigment. The furnace used for the purpose is shown in Figs. 116 and 117.

In both figures, (1) represents a pair of cast-iron cylinders, 10 ft. long and 12 in. in diameter, (2) a heavy spiral screw with four longitudinal flat-iron bars (not shown), the screw conveying the fume from feed to discharge and compressing it, the bars raising it and showering it through the air current passing through the cylinder, (3) a driving gear, (4) the feed-hopper, (5) the drum for collecting fine dust, (6) the screw conveyer for refined pigment, (7) the

[1] Upon heating, $PbSO_3$ is decomposed into $PbSO_4$, PbO, and SO_2.

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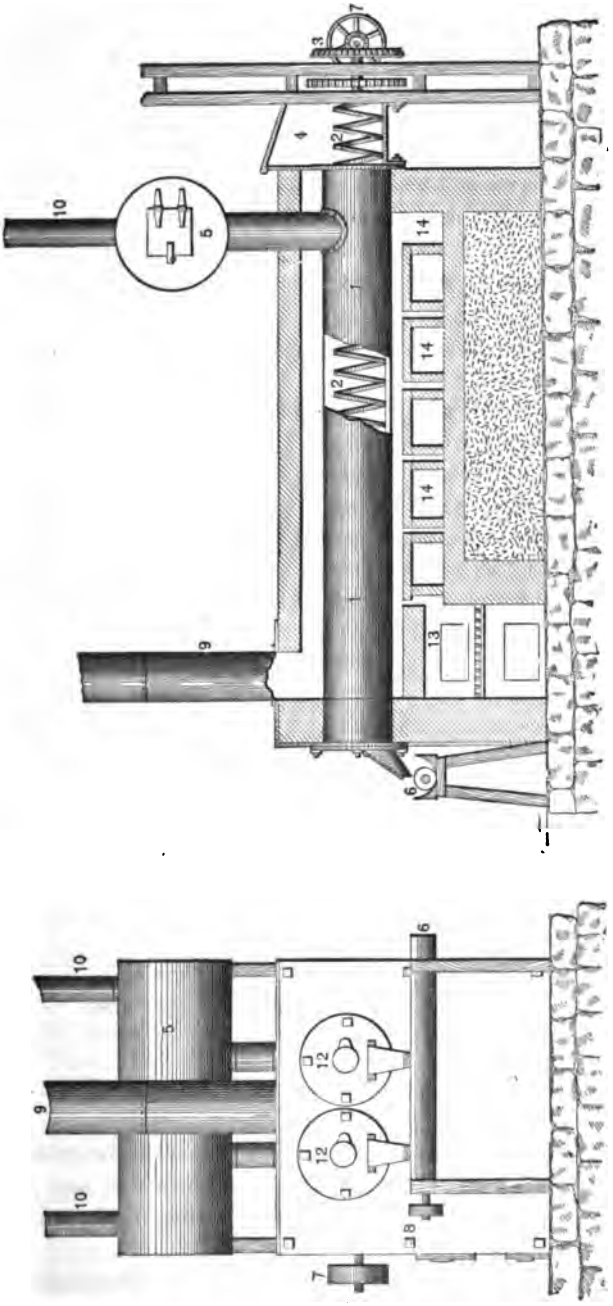

FIGS. 116 and 117.—Bartlett refining furnace for making zinc-lead pigment.

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driving pulley, (8) the pulley of screw conveyer, (9) the chimney for products of combustion, (10) the chimney to carry off gases from the drum, (12) a cast-iron disk with circular air-inlet and discharge-spout, (13) the fireplace, and (14) flues for the products of combustion. Four cylinders refine all the fume of the plant, a cylinder treating from 1200 to 1500 lb. of fume per day. The temperature of the cylinders is kept at about 815° C. The pigment, passing in about 20 min. through a cylinder, is discharged at (6) and then passed through bolting cloth which screens out any coarse particles. Four volumes of raw fume give one volume of refined pigment. This consists mainly of 54–46 per cent. ZnO, 40–50 per cent. $PbSO_4$, a small amount of $ZnSO_4$ and an excess of O, the ultimate analysis giving Zn 47.33, Pb 24.92, S 2.96, Fe_2O_3, etc., 0.45, O 24.34 per cent.

It is bluish-white (white lead being yellowish-white), dense, and sells at 1 ct. a pound less than white lead.

The amount of dust collected in the drum (5) is small (1 lb. per ton of ore); it contains many rare elements.

The plant of the American Zinc-Lead Co., of Cañon City, Colo., contained 12 sintering furnaces (six of which were always running), three blast furnaces (one being held in reserve), and treated about 100 tons of ore per day. The power required per ton of ore was about 3.25 h.p., and the labor 0.75 man. The power was distributed as follows: Five exhaust fans (6 ft.), 175 h.p.; three underwind fans, 40; two Baker blowers, 30; one crusher and roller plant, 50; one electric light plant, 20; one refining plant, 10—or a total of 325 h.p.

The cost of treatment was not made public, but a treatment charge for galena-blende concentrates published by Bartlett[1] gives an approximate idea. His offer was:

21.2 oz. silver at 68.5 ct. per oz., less 5 per cent.		$13.78
0.05 oz. gold at $19 per oz.		0.90
18 per cent. lead at 25 ct. per unit		4.50
26.8 per cent. zinc, nothing		0.00
Gross value		$19.18
Less treatment	$5.80	
Less freight	5.20	
		11.00
Net value		$8.18

The plant was shut down about 1908,[2] as it ceased to be profitable because at that time electromagnetic concentration came into use, and payments had to be made for the zinc in the ore instead of having the benefit of the usual deductions for its presence.

With the advent of the economic deposition of zinc from sulphate solution, the Bartlett process was put into a position to recover the zinc from the vola-

[1] *Eng. Min. J.*, 1896, LXI, 587.

[2] Jackling, *Min. Sc. Press*, 1917, CXIV, 617.

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tilized oxide fume and thereby enabled to pay for part of the zinc of the ore. This led to the installation of a new plant at Florence, Colo., in 1915–16.[1]

Following the example of Bartlett, mixed zinc-lead sulphide ore was smelted in a water-jacket blast-furnace 42 by 156 in. at tuyères with a working height of 5 ft. The ore averaged Zn 20, Pb 5, Cu 1, S 20–25 per cent., and contained Ag 6–8 oz. per ton and some Au. The percentage of coke used was high. The slag analyzed SiO_2 38, CaO 22–25, ZnO 7 per cent., the rest being mainly FeO; it ran low in Pb, contained Cu 0.2 per cent., and was free from Ag. The matte-fall was 5 per cent. The fume ($PbSO_4$+ZnO) retained some Zn_2SO_4, the total S was about 7 per cent.; the Ag-content, which was generally proportionate to that of Pb, ranged from 7 to 10 oz. per ton. The campaign of a furnace was from 3 to 5 weeks. The fume was shipped to Keokuk, Ia., and leached with H_2SO_4; the dissolved Zn was recovered by electrodeposition, and the insoluble residue smelted for lead bullion.

The large percentage of high-price coke required, the cost of labor, and the difficulties encountered in smelting were the cause of a change from blast furnace to reverberatory smelting. At present the ore is crushed, rough-roasted in a Wedge kiln, and the roasted ore, mixed with coal, smelted in a dust-fired reverberatory furnace for matte, slag, and fume. The slag made assays SiO_2 from 30 to 50 per cent.; it retains about 2 per cent. S, and less Ag than that from the blast furnace, averaging about 0.3 oz. per unit of Pb. The fume is treated as shown above.

[1] R. G. Hall, private communication, March, 1918.

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CHAPTER IX

SMELTING LEAD ORES IN THE BLAST FURNACE[1]

74. Introduction.—The process carried on in the blast furnace is one of reduction. The aim of smelting a lead ore is to set free the lead of the ore, to collect other metals such as Cu, Ni, etc., in a matte, and to form of the gangue a slag low enough in values to be a waste product. Precious metals which enter mainly the lead, but to some extent also the matte, are recovered by separate processes.

All lead ores can be smelted in the blast furnace; any ore containing over 4 per cent. SiO_2 must be so treated. If the lead ore is a sulphide, it usually is first roasted or blast roasted; if a carbonate or a mixture of carbonate and sulphide, the carbonate prevailing, it is smelted at once. A study of the chemical analyses of argentiferous ore, given in §49 and 50 will show that these ores usually contain much over 4 per cent. SiO_2. This explains the universal use of the blast furnace in smelting the ores of the Rocky Mountains and Pacific divisions of this country. In Table 41 are listed the silver-lead smelting works of the United States, Canada, and Mexico in operation in 1917.[2]

With the low-SiO_2 non-argentiferous lead ores of the Mississippi Valley, which have not been sufficiently enriched by mechanical concentration to furnish a product suited to the reverberatory furnace or the ore-hearth, the blast furnace is the only efficient smelting apparatus.

The treatment of lead ores in the blast furnace used to be, and sometimes still is, discussed under the heads of Precipitation and Roasting and Reduction for sulphide ores, and of General Reduction for carbonate ores. This classification, which had its origin on the European continent, was justified by the practice of important smelting centers in which large mines furnished smelteries with uniform ores which were smelted in one of the three ways. Thus, *e.g.*, in the Harz Mountains, Prussia, galena concentrates used to be smelted with the addition of sufficient iron to carry on the so-called precipitation process. In other parts of the European continent silicious sulphide ores were ordinarily subjected to slag-roasting in a hand reverberatory furnace and the product smelted in the blast furnace; there prevailed the roasting and reduction process. The treatment of oxide ores in Europe is confined to intermediary products of lead refineries, and these are rarely smelted by themselves; hence the general reduction process was carried out only in exceptional cases.

In the centrally situated custom smelteries of the United States, which obtain their ores from different parts of the country in varying quantities, the

[1] Dwight, *Eng. Min. J.*, 1916, CII, 671.

[2] *Eng. Min. J.*, 1918, CV, 63.

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TABLE 41.—AMERICAN SILVER-LEAD SMELTING WORKS, 1917

Company	Place	Furnaces	Annual capacity (a)
American Smelting and Refining Co.	Denver, Colo.	7	510,000
American Smelting and Refining Co.	Pueblo, Colo.	7	380,000
American Smelting and Refining Co.	Durango, Colo.	4	210,000
American Smelting and Refining Co.	Leadville, Colo.	10	510,000
American Smelting and Refining Co.	Murray, Utah	8	657,000
American Smelting and Refining Co.	East Helena, Mont.	4	306,000
American Smelting and Refining Co.	Omaha, Neb. (b)	2	82,000
American Smelting and Refining Co.	Perth Amboy, N. J. (b)	4	170,000
Con. Kansas City Sm. and Ref. Co.	El Paso, Tex.	6	380,000
Bunker Hill & Sullivan Min. and Concentrating Co.	Kellogg, Ida.	3	600,000
Selby Smelting and Lead Co.	Selby, Cal.	3	210,000
Ohio & Colorado Smelting Co.	Salida, Colo.	4	345,000
United States Smelting Co.	Midvale, Utah	7	530,000
Northport Smelting and Refining Co.	Northport, Wash.	2	216,000
Pennsylvania Smelting Co.	Carnegie, Penn.	2	60,000
International Smelting Co.	Tooele, Utah	5	600,000
Totals, United States		78	5,766,000
American Smelting and Refining Co.	Monterey	10	584,000
American Smelting and Refining Co.	Aguascalientes	1	40,000
American Smelting and Refining Co.	Chihuahua	7	400,000
American Smelters Securities Co.	Velardeña	3	150,000
Compañia Metallurgica Mexicana	San Luis Potosi (c)	10	250,000
Compañia Metallurgica de Torreon	Torreon	8	360,000
Compañia Minera de Penoles	Mapimi	6	325,000
Totals, Mexico		45	2,109,000
Consolidated Mining and Smelting Co.	Trail, B. C.	4	140,000

(a) Tons of charge. (b) Smelts chiefly refinery intermediary products. (c) Not being operated, but plant is expected to start in the near future.

regularity of treatment prevalent in Europe cannot be followed. All the ores received go to make up the blast furnace charges; they are variously apportioned in order to form suitable mixtures as regards lead-contents and the amounts and kinds of slag- and matte-forming constituents. The sulphide ores may or may not be roasted or blast roasted; oxide ores will be smelted raw without any preliminary treatment unless they are too fine for this purpose, and are better suited to make up blast roasting charges. In order to meet these conditions, smelting in the blast furnace will be treated broadly as a fusion in which reduction and scorification are the leading chemical processes, while precipitation and sulphurization are of secondary importance.

In view of the fact that the principal metal-bearing minerals which enter the blast furnace are sulphides, and that they usually have to be roasted or blast roasted before they are used to make up the charges, it is necessary to consider first these two preparatory operations.

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75. Oxidizing Roast of Metallic Sulphides in Powder Form.—The general reactions that take place when a metallic sulphide in powder form is heated with access of air have been treated elsewhere.[1] Here the behavior of the several sulphides likely to be met in lead smelting will be studied.

1. *Lead Sulphide, PbS.*—The behavior of PbS has been studied in §39–41.

2. *Iron Monosulphide, FeS.*—The changes FeS undergoes may be expressed according to Plattner[2] by: $FeS+3O=FeO+SO_2$; $3FeO+O=Fe_3O_4$ and $SO_2+O+\text{catalyzer}=SO_3$; $2Fe_3O_4+SO_3=3Fe_2O_3+SO_2$ and $FeO+SO_3=FeSO_4$; $2FeSO_4+\text{heat}=Fe_2SO_6+SO_2$ and $Fe_2SO_6+\text{heat}=Fe_2O_3+SO_3$.

Pure FeS ignites in air at from 325° (0.1 mm. or 100-mesh) to 472° C. (>0.2 mm. or 50-mesh) and does not decrepitate upon heating.[3] Heating $FeSO_4$ in a current of air[4] causes it to be readily decomposed at 480° C. forming yellow-brown $Fe_2O_3.2SO_3$, and this basic ferric salt is completely changed at 560° into red Fe_2O_3 and SO_3. The basic salt $Fe_3S_2O_9$ of recent investigation in the laboratory differs from the Fe_2SO_6 of Plattner; in the furnace both are likely to be formed.

3. *Iron Disulphide, FeS_2.*—If heated with exclusion of air to 200° C.,[5] S begins to distil off; at 700° the dissociation into FeS and S is complete.[6] Heating with access of air[7] to 250°–290° C. causes oxidation take place according to $FeS_2+3O_2=FeSO_4+SO_2$. At temperatures lying between 290 and 500° the reaction $4FeS_2+11O_2=2Fe_2O_3+8SO_2$ prevails. Pyrite[8] from Elba gives off SO_2 at 450° C. and glows at 533°; that from Rio Tinto[9] begins to roast at 260°–275° C.

Although FeS_2 contains more S than FeS, it roasts with greater rapidity because in the process some S is distilled off which burns freely and leaves porous the remaining FeS, which now offers many points of attack for oxidation; there is also set free a larger amount of SO_3, which has a strongly oxidizing effect.

4. *Cuprous Sulphide, Cu_2S.*—Oxidation may be expressed according to Plattner[10] by $Cu_2S+3O=Cu_2O+SO_2$; $Cu_2O+2SO_2+O\rightleftarrows 2CuO+SO_3$; $CuO+SO_3\rightleftarrows CuSO_4$. Laboratory experiments by Aubell[11] show that prepared Cu_2S begins to roast at 200° C. with the reaction $2Cu_2S+5O_2=2CuO+2CuSO_4$, which continues up to 330; that above 330° the reaction $Cu_2S+O_2=2CuO+SO_2$ begins; and that up to 550° more than half of the sulphide-S is converted into sulphate-S. He finds also that SO_3, set free by the dissociation of $CuSO_4$,[12] acts

[1] Hofman, H. O., "General Metallurgy," McGraw-Hill Book Co., New York, 1918, 403.
[2] Plattner, C. F., "Die Metallurgischen Röstprocesse," Engelhardt, Freiberg, 1856, 133.
[3] Friedrich, *Metallurgie*, 1909, VI, 169.
[4] Hofman-Wanjukow, *Tr. A. I. M. E.*, 1912, XLIII, 548.
[5] Kothny, *Oester. Jahrb.*, 1910, LVIII, 112; *Metallurgie*, 1911, VIII, 389.
[6] Geodel, *J. für Gasbeleuchtung*, 1905, XLVIII, 400.
Friedrich, *Stahl u. Eisen*, 1911, XXXI, 2040.
[7] Kothny, *loc. cit.*
[8] Friedrich, *loc. cit.*
[9] Chalon, *Rev. Un. Min.*, 1902, LVII, 201.
[10] *Loc. cit.*
[11] *Oester. Jahrb.*, 1910, LVIII, 131.
[12] $2CuSO_4$ at 670° C. $=2CuO.SO_3+SO_3$; $2CuO.SO_3$ at 736° C. $=2CuO+SO_3$.
Hofman-Wanjukow, *Tr. A. I. M. E.*, 1912, XLIII, 547.

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upon Cu_2S forming Cu_2O as seen by $Cu_2S+3SO_3=Cu_2O+4SO_2$, and thus explains the formation of Cu_2O in the ore as long as there is present any undecomposed Cu_2S. Plattner was aware of the effect of SO_3 upon Cu_2S as well as upon Cu_2O, expressed by $Cu_2O+SO_3=2CuO+SO_2$; he differs from Aubell in assuming the formation of Cu_2O from CuO, which the latter omits. Chalcocite, from 0.1 to >0.2 mm. or 100- to 50-mesh in size, ignites in air between 430 and 697° C.,[1] and does not decrepitate.

5. *Cupric Sulphide, CuS.*—The behavior of CuS is similar to that of Cu_2S, as the former readily gives up one atom of S when heated to a bright-red with exclusion of air; by analogy with FeS_2 and FeS it ought to roast more readily than Cu_2S.

6. *Chalcopyrite, $CuFeS_2$.*—This common copper mineral shows a behavior similar to that of FeS except that it decrepitates readily upon heating; there are formed Fe_xO_y and $FeSO_x$ as well as Cu_xO_y and $CuSO_x$.

7. *Zinc Sulphide, ZnS.*—The oxidation of blende may be expressed by $2ZnS+7O=ZnO+ZnSO_4+SO_2$. The normal sulphate, $ZnSO_4$ is converted[2] at 720° C. into white $3ZnO.2SO_3$, and this at 767° into ZnO and SO_3, the ZnO being yellow when hot and white when cold. Decrepitation and ignition temperatures vary with the character of the mineral. Friedrich[3] found that Spanish blende decrepitated at 40° C., Hofman[4] ascertained that blende from Warren, N. H., did not decrepitate at all, whereas this was decidedly the case at 290° with samples of blende from Joplin, Mo., and New Mexico. The blende of Friedrich with 0.36 per cent. Fe, when 0.1 mm. or 100-mesh in size, ignited at 647° C.; when >0.2 mm. or 50-mesh, this took place at 810°. The Joplin blende with 0.45 per cent. Fe ignited at 480° C.; the Warren blende with 8.80 per cent. Fe at 557°, when through 8- and on 12-mesh, and at 515° when through 80-mesh; the blende from New Mexico with 13.4 per cent. Fe ignited at 534°. It appears therefore that a ferruginous blende has a higher ignition temperature than one which runs low in Fe. The earlier investigations of Minor[5] and Lepiarezyk[6] bear out this deduction. The presence of Fe appears also to retard a dead-roast. Whatever may be the character of the blende from different localities, one fact stands out clearly, that the mineral roasts very slowly and demands considerable time for complete desulphurization; it requires in addition an abundance of air, a red-heat at the start, and a deep-orange toward the finish. On account of the length of time necessary to obtain a satisfactory roast, and the short time given to the operation of blast roasting, it has been found that in blast roasted sulphide ore a large part of the original blende has not been decomposed at all, and subsequently causes trouble in the blast furnace.

[1] Friedrich, *Metallurgie*, 1909, VI, 169.
[2] Hofman-Wanjukow, *Tr. A. I. M. E.*, 1912, XLIII, 523.
[3] *Metallurgie*, 1909, VI, 167.
[4] *Tr. A. I. M. E.*, 1905, XXXV, 811.
[5] *Chem. Z.*, 1889, XIII, 1602; *Berg. Hüttenm. Z.*, 1889, XLVIII, 466.
[6] Thesis, Kgl. Tech. Hochschule, Berlin, 1908.

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8. *Manganese Sulphide, MnS.*—The mineral albandite is converted by roasting into $MnSO_4$ and Mn_3O_4; the $MnSO_4$ is readily converted at 790° C.[1] into dark red to black Mn_3O_4, while SO_3, SO_2, and O are set free. The ignition temperature was found by Friedrich[2] to lie at 355° C., when 0.1 mm. or 100-mesh in size, and at 700°, when >0.2 mm. or 50-mesh. The mineral does not decrepitate.

9. *Silver Sulphide, Ag_2S.*—If argentite is given an oxidizing roast, it is converted into finely divided Ag and SO_2 as seen by $Ag_2S+O_2=Ag_2+SO_2$. The compound Ag_2O cannot form, as it is dissociated at 383° C.[3] into Ag_2 and O. The ignition temperature of argentite, 0.1 mm. or 100-mesh in size, was found by Friedrich[4] to be 605° C., and 875° if of 0.2 mm. or 50-mesh size. The roasting of Ag_2S is attended by loss in Ag. If present in small quantities in other metallic sulphides, the SO_3 set free in their decomposition has a tendency to exert a sulphatizing effect: $2Ag+SO_3=Ag_2SO_4+SO_2$ and $Ag_2S+4SO_3=Ag_2SO_4+4SO_2$. The decomposition of Ag_2SO_4 whether caused by heat or the presence of oxides, such as Fe_3O_4 or Cu_2O, is attended by a considerable loss in Ag. The reaction may be expressed by Ag_2SO_4 at 925° C. $=Ag_2+SO_3+O$;[5] $Ag_2SO_4+4Fe_3O_4=2Ag+6Fe_2O_3+SO_2$; $Ag_2SO_4+2Cu_2O=2Ag+4CuO+SO_2$. In roasted ore the silver will be found to be present as undecomposed Ag_2S, as Ag_2SO_4, and as Ag.

76. Roasting, General.[6]—The aim of an oxidizing roast is to oxidize metallic sulphide and to drive off as much S as possible in the form of SO_2.

The Hall process,[7] which aims to roast sulphide ore in such a way as to convert the metal into oxide, to expel the S as vapor, and to recover the vapor in the form of flower of sulphur, has not yet been tried in connection with lead ores.

A question to be decided with each ore is whether it is necessary to roast it, or it is better to smelt it raw. The character and amount of sulphide, the richness of the ore, and the cost of the operation will be the deciding factors. As a rule, any ore containing 8 per cent. S or over is best roasted before it is smelted. The richness of the ore in Ag and its coarseness may modify this general rule. As roasting is always connected with loss in metal, it may be more profitable to smelt raw an ore rich in Ag even if it contains over 8 per cent. S, than to first give it an oxidizing roast. A lead ore with 100 oz. Ag per ton is rarely roasted; some metallurgists[8] have drawn the line at 50 oz. per ton, which however seems to be rather low.

Gold mill concentrates with <8 per cent. S are practically always roasted, as the common sulphide mineral pyrite, FeS_2, is readily oxidized, and the Fe_2O_3 freed from S forms a valuable flux; further, being finely divided, the

[1] Hofman-Wanjukow, *Tr. A. I. M. E.*, 1912, XLIII, 523.
[2] *Loc. cit.*
[3] Stahl, *Metallurgie*, 1907, IV, 690.
[4] *Loc. cit.*
[5] Hofman-Wanjukow, *loc. cit.*
[6] These general considerations hold good for blast roasting, §84.
[7] Hofman, *Min. Ind.*, 1914, XXXIII, 477.
[8] Newhouse, *Eng. Min. J.*, 1891, LI, 260.

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concentrates are likely to cause trouble in the furnace unless they are somewhat agglomerated.

Pure argentiferous galena ores do not often come to smelteries in such quantities as to make a separate treatment necessary; striking exceptions are found in some works in Idaho and Montana. Ordinarily, pure galena concentrate is added to an ore-bed in such quantity as to make up for the common deficiency in lead of the blast furnace charge. Impure galena ore is usually mixed with sulphide ores free from lead before roasting. This is done for two reasons, to reduce the S-content of the roasting charge, and to furnish the required lead for the blast furnace charge. If a roasting charge contains over 20 per cent. S, it is likely to become sticky at the beginning of the roast, and this must be avoided if the S is to be eliminated satisfactorily.

Pyrite ores in the blast furnace consume much S as seen by $FeS_2+Fe=2FeS$, they increase the cost of smelting, reduce the capacity of the furnace for ore, and form much matte which, on account of its content of Pb and Ag, diminishes the direct yield of these two metals in the lead bullion. The matte has to be roasted before it is resmelted to recover these values. The Fe originally added to the ore-charge becomes again available, so that the actual consumption of Fe in smelting raw sulphide ores is not so great as is generally assumed. Whether a pyritic ore shall be roasted or not is decided by the precentage of S it contains over that which is required to cover as Cu_2S the amount of Cu present. However, a large matte-fall has to be avoided in smelting, as the amount of Ag entering the slag increases with the percentage of matte formed. This bad effect begins to be seen with 10 per cent. of matte in the charge; every furnace man tries to keep his matte-fall under 5 per cent.

Blende has a very deleterious effect upon blast furnace work by forming a mushy matte, which interferes with the work at the tuyère-level, and by incrusting the sides of the shaft with fumes of zinc, more or less oxidized, resulting from the decomposition of ZnS by Fe or Cu, and the subsequent volatilization of the Zn. If present to any extent, the ore will have to be roasted so that the oxide may be taken up by the slag and thus carried out of the furnace. In a general way it may be said that the higher the percentage of Pb in the charge, the greater is the amount of ZnS that is permissible. For instance, if in the practical absence of FeS_2, the percentage of Pb is twice that of Zn, the ore may be smelted raw; if Pb and Zn are present in equal amounts, or if there is less Pb than Zn, the ore must be roasted before smelting. Attention has been called in §75(7) to the slowness with which ZnS gives off its S.

77. Roasting Furnaces, General.—The roasting of lead ores can be carried on in heaps, stalls, kilns, and reverberatory furnaces. So-called mixed sulphide ores, consisting mainly of galena, pyrite, chalcopyrite, and blende, are sometimes roasted in heaps and stalls, the sulphurous gases being allowed to go to waste; or they are roasted in kilns, when the SO_2 is to be converted into H_2SO_4. As this manner of roasting is comparatively rare with lead ores, but common with copper ores, and as the apparatus is practically the same, the methods are omitted here. They have been fully discussed

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elsewhere.[1] One instance, the heap-roasting of concentrator slimes at Port Pirie, N. S. W.,[2] deserves mention, as it is a simple and inexpensive method of handling a very troublesome material, if it cannot be added to the charges of the blast roasting apparatus.

Before the advent of blast roasting the reverberatory furnace was pre-ëminently the apparatus in use for roasting lead ores. Hand-rabbled furnaces of different forms were in operation in every smeltery for the roasting of ore and matte. The great success of mechanical furnaces in the rough-roasting of sulphide copper ores prompted lead works to follow the example of copper metallurgists with the result that the mechanical roasters of Ropp, Pearce, Brown, O'Hara, Keller, Wethey, and Brückner, were erected and adapted to the treatment of lead ores. Though they reduced the cost of roasting in comparison with the work in the hand-reverberatory furnace, their work on the whole was not satisfactory for several reasons.

In order to reduce the S-content of a lead-bearing ore to a required degree, it is, if not necessary, at least desirable to be able to accelerate and retard the travel of the ore from the feed to the discharge, and toward the end of the roast, when the temperature is brought to the highest permissible point, to rake and turn over the charge much more frequently than at the beginning, if a satisfactory elimination of S is to be effected. In a mechanical furnace with stationary hearth the raking and turning-over of ore is uniform, as it depends upon the speed of the rakes. If these travel too slowly, the ore will be insufficiently worked near the discharge; if too quickly the ore near the feed will be turned under while the S on the surface is still burning, with the result that the ore will cake and form lumps. As this must be avoided, the mechanical furnace has to be run slowly, and the ore not being sufficiently roasted toward the end of its path, will retain a comparatively large amount of S. The relatively small quantity of ore the mechanical furnace puts through when it has to reduce the S-content to say 3 per cent., very often does not warrant the expense of erecting a furnace, and of keeping it in repair.[3] Another difficulty is that of the ore caking on the bed of loose material with which the brick hearth is covered, and obstructing the passage of the stirrers. This has been overcome in part by plowing up a crust as soon as it forms; others have found it preferable to allow a crust to form and wear off the blades of the rabbles, and to shut down the furnace when the blades become too short, remove the crusts, and supply new blades to the arms. It has also been found that the blades last a much shorter time in roasting lead ores than pyritic ores. This must be caused by a chemical effect of the ore, as there is no reason why the mechanical wear should be greater with lead-bearing than with copper-bearing

[1] Hofman, H. O., "Metallurgy of Copper," McGraw-Hill Book Co., New York, 1918, 71–114.

[2] Horwood, *Tr. Austr. Inst. Min. Eng.*, 1903, IX,[1] 106.
Delprat, *op. cit.*, 1907, XII, 15; *Eng. Min. J.*, 1907, LXXXVII, 317, 517.
Hofman, H. O., "General Metallurgy," McGraw-Hill Book Co., New York, 1918, 635.

[3] Iles, *Eng. Min. J.*, 1900, LXX, 185.

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ore. It has been suggested that PbS may be decomposed to some extent by the red-hot blades. Should this be the case, the loss in Pb and Ag by volatilization in a mechanical furnace should be greater than in the hand reverberatory and in a furnace with revolving hearth, but no evidence to that effect has been recorded.

In a mechanical furnace having a revolving hearth, of which that of Brückner is the only representative, the ore can be turned over quickly or slowly by varying the number of revolutions; further, it can be kept any length of time exposed to the oxidizing heat of flame and gases. The disadvantages of the furnace, such as lack of air, small tonnage, high fuel consumption, adhesion of ore to walls, balling of ore, and quantity of flue-dust formed, have militated against an extended use.

Though some mechanical reverberatory furnaces are still found in operation, most of them have been scrapped, and new ones will not be erected any more. It is therefore superfluous to take up the details here, especially as they have been fully treated elsewhere.[1]

78. Oxidizing Roast of Metallic Sulphide in Reverberatory Furnace.—The results obtained in roasting a powdery sulphide ore in a reverberatory furnace depend not only upon the mineralogical character of its components, but also upon the size to which the ore has been crushed, the thickness of its bed upon the hearth, the amount of rabbling it receives, the time it remains in the furnace, and the temperature to which it is exposed.

Ignition temperatures of metallic sulphides of different sizes have been studied by Friedrich.[2] He found that 12-mesh material will give off SO_2 and show incandescence at a lower temperature than 6-mesh; also that pyrite, pyrrhotite, galena, and blende will show similar phenomena with a rising temperature in the order in which they have been named. The inferences from these laboratory experiments agree with those from practical experience.

As galena oxidizes slowly when heated with access of air, a large number of surfaces (fine-crushing) is necessary, if the roast is to have the desired result. Then, as the roast progresses in each particle from the surface to the center, it is probable that if the particles are too coarse, a reaction will take place at the points at which PbO and $PbSO_4$ formed are in contact with undecomposed PbS, and the oxidation of the resulting Pb will cause a considerable loss of Pb and Ag —another reason for fine-crushing. Ores that do not roast readily, that is, ores rich in galena and blende are crushed to pass an 8-mesh sieve. Ores that roast readily, *e.g.*, pyritic ores and iron matte with about 10 per cent. Pb, are crushed through a 4-mesh sieve. The oxidation of these is rapid, and the roasted product less fusible and more porous than it would be if richer in lead.

The thickness of the charge on the hearth and the amount of necessary rabbling depend also upon the character of the ore. The richer it is in Pb and Zn, the thinner must be the layer, and the more frequent the rabbling. The

[1] Hofman, H. O., "Metallurgy of Copper," McGraw-Hill Book Co., New York, 1918, 116.

[2] *Metallurgie*, 1909, VI, 169.
Hofman, "General Metallurgy," 1918, 404.

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thickness of bed varies from 3 to 6 in.; the rabbling is repeated every ¾ to 1½ hr.

The time required to roast an ore depends upon the readiness with which it oxidizes and fuses. Ores in which galena prevails require a slow roast and a low temperature throughout, as even with the most careful work it is difficult to prevent roasted ore from retaining undecomposed PbS. Pyritic ores can be roasted quickly; there is no danger of half-roasted ore becoming sticky and adhering to the hearth of the furnace. Ores rich in blende require a considerable time and a comparatively high temperature if the ZnS is to be completely converted into ZnO and $ZnSO_4$, and the latter to be fully decomposed.

The temperature at which the roasted ore is to be withdrawn from the furnace must be regulated by the character of the sulphates formed and by the fusibility of the charge. The charge can be taken out of the furnace as a pulverulent, a sintered, or a fused mass. As regards subsequent smelting, it is best to slag the ore, as by obtaining the roasted ore in lump form the disadvantages of treating fine ores in the blast furnace are overcome; but other considerations prevent this. The leading ones are loss in Pb and Ag, and the increase in cost. Newhouse[1] gives as a result of a series of experiments in roasting ores containing from 12 to 18 per cent. Pb, a loss of from 15 to 18 per cent. Pb and of 2 per cent. Ag with fusion of roast, and one of from 2 to 5 per cent. Pb with no loss in Ag without fusion.

In Missouri with pure galena concentrate the loss in Pb used to be 10 per cent. with fusion. By agglomerating or sintering the roast instead of fusing, the loss in metal will be only slightly higher than when it remains pulverulent. The S will not be so effectually removed as when the ore is slagged, but more so than when it remains a powder. For instance, slag-roasted ore contains from 1 to 3 per cent. S, while roasted pulverulent ore contains from 3 to 7 per cent. S. The loss increases on the whole with the lead-content of the charge. It used to be said that an ore with 10 per cent. Pb or less could be safely slagged; with from 10 to 20 per cent. Pb only agglomerated; with over 20 per cent. Pb the temperature had to be kept so low that the roasted product remained pulverulent, or only slightly adherent when withdrawn from the furnace. Such ore was drawn from the furnace into a slag-pot and pounded hot with iron disks so as to make the particles adhere to one another.[2]

These statements refer to mixed sulphides formerly treated in Western smelteries, which as a rule ran low in Pb and high in Ag. They are not applicable to pure galena concentrates practically free from Ag (as in the Mississippi Valley) or low in Ag (as in most European silver-lead works), because in both cases the ores used to be slagged to a greater or less degree. The charges, running from 50 to 60 per cent. Pb and being nearly free from impurities, require, when roasted, only a very slight increase in temperature to be slagged,

[1] *Eng. Min. J.*, 1891, LI, 260.

[2] Iles, *Eng. Min. J.*, 1900, LXX, 184.

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care being taken to keep the temperature as low as possible. Therefore the loss in Pb and Ag is not great although the percentage of Pb is high.

79. Long-bed Reverberatory Hand-roasting Furnace.—This old standby has also seen its best days; it still retains, however, its usefulness in cases in which blast roasting has not been able to make headway. The furnace has a single roasting hearth, from 40 to 80 but usually 60 ft. long and from 14 to 17 but usually 16 ft. wide, a fireplace at one end and a flue at the other, and working doors on either side through which the ore, fed near the flue-end, is slowly worked down to the bridge-end, and there discharged.

Furnaces with two superposed hearths[1] have been used in order to save floor-space and to reduce the fuel consumption. The disadvantages, such as more solid construction, repair on lower hearth requiring removal of upper hearth, difficulty of moving ore on upper hearth while standing on a shaky platform with resulting carelessness of work and building up of crusts which have to be burnt out or cut out, etc., have made the double hearth the exception. Loss of heat through the roof of the single hearth is easily corrected by covering with sand; in the double-bed furnace the heat passing off through the roof of the lower hearth heats the floor of the upper.

Three kinds of single-hearth furnaces have been erected in lead smelteries: A furnace having a level hearth, a slagging hearth (fuse-box), and a sintering hearth. Though the level hearth is still in use, and the fuse-box was replaced in Colorado about 1898[2] by the sintering hearth, and the last has made way in most plants to blast roasting apparatus, the three types have not outlived their usefulness, and deserve to be discussed.

80. Reverberatory Hand-roasting Furnace with Level Hearth.—This furnace, shown in Figs. 118–123, was in operation at the works of the Montana Smelting Co., Great Falls, Mont. The hearth, 40 by 14 ft., is shorter than is usually the case, the general length being 60 ft. The reason for the shortness is that the charge is to be withdrawn near the firebridge in a pulverulent state through two discharge-openings in the bottom; the temperature being kept low at the firebridge, the length of 40 ft. is sufficient to utilize all the heat generated on the grate. Attention may be called to the vaulted arches which support the hearth; the absence of offsets in the hearth, which has a gentle rise from bridge to flue, while with 2-in. offsets every 10 ft., there would be four level hearths separated from one another by 2-in. steps; the slope from center-line to sides; the discharge-openings for roasted ore; the damper in the flue; the distance of 54 in. between working doors, which is less than usual (72–96 in.); the width of fireplace, 21 in., Fig. 123, which, proving insufficient for the low-grade coal used, was increased to 36 in., Fig. 118; to the narrowing of the firebridge from 27 in., Fig. 123, to 22.25 in., Fig. 118, to furnish the room necessary for the en-

[1] Mechernich, *Berg. Hüttenm. Z.*, 1875, XXXIV, 129 (Jaeger); *op. cit.*, 1886, XLV, 434; *Eng. Min. J.*, 1877, XXIII, 136 (Huppertz).

Přibram, *Oester. Jahrb.*, 1890, XXXVIII, 11 (Zdráhal).

[2] Iles, *Eng. Min. J.*, 1900, LXX, 184.

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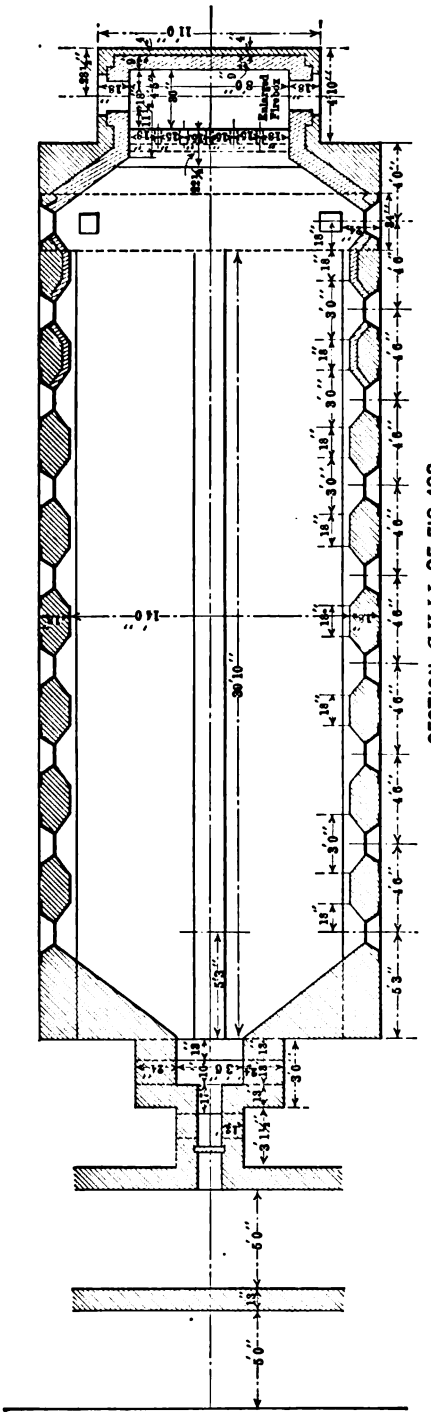
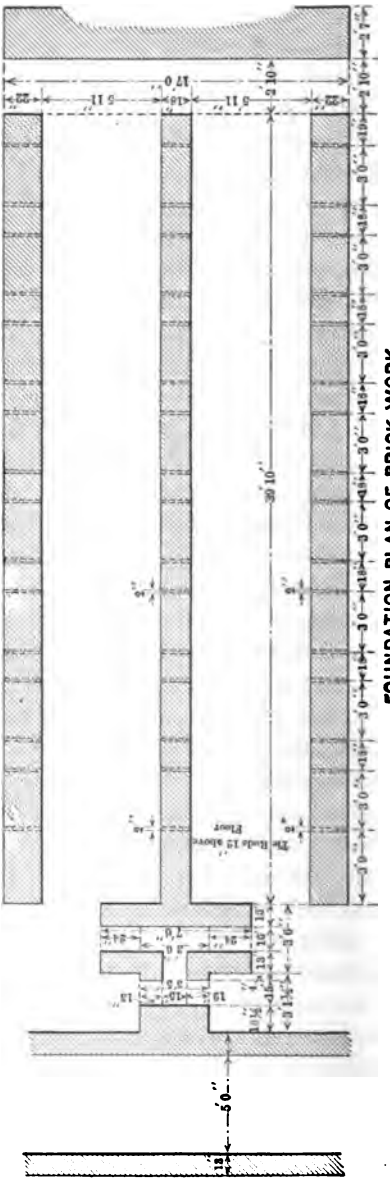


SECTION G H, I J, OF FIG. 123
FIG. 118



FOUNDATION PLAN OF BRICK WORK
FIG. 119

FIGS. 118 and 119.—Reverberatory hand-roasting furnace with level hearth.

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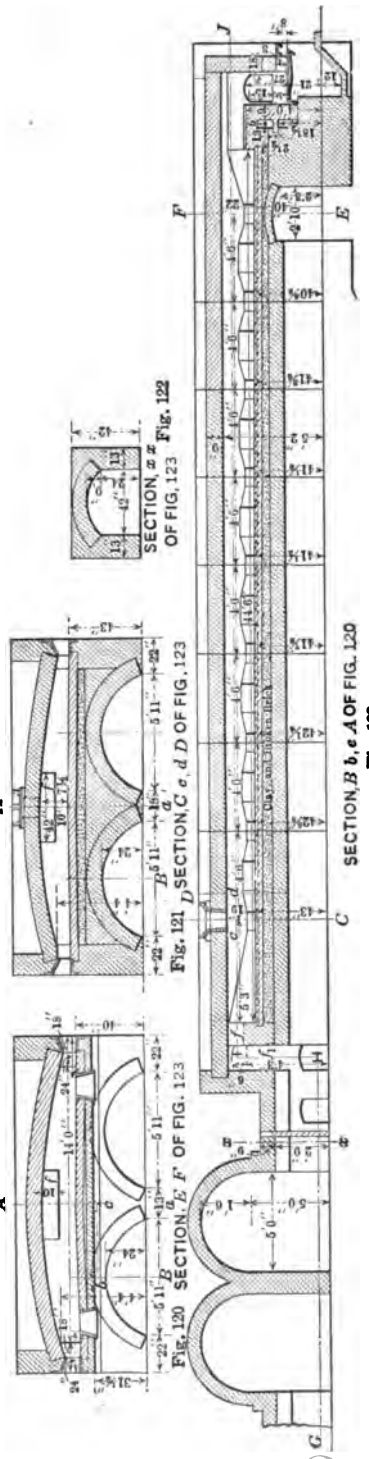
largement of the fireplace; and the addition of air-flues in the firebridge to furnish additional air for oxidation.

The figures show that the hearth has a uniform thickness of 9 in., that is, the brick have been laid on end. This is not necessary in parts of the furnace in which the charge does not become sticky; the brick can be laid on edge, that is, the hearth can be 4.5 in. thick. Near the fire-end fire-brick are always used; in the other parts strong red bricks are sufficient to withstand heat and mechanical wear.

The horizontal grate of the furnace is replaced in many works by a step-grate when non-caking bituminous coal or lignite serves as fuel. In recent years oil has replaced solid fuel[1] in localities where it is sufficiently low-priced; the excess air required for the atomization of oil over that which is necessary for combustion, and the in-rush of air around the nozzles furnish a strongly oxidizing atmosphere. Producer gas is used in a few works;[2] it would have made better headway than it has, were it not for the diminished number of furnaces in operation at present.

As regards furnace-equipment, attention may be called to the automatic charging device by Löwe[3] which used to do very satisfactory work at Přibram, Bohemia. A furnace received six charges in 24 hr.; charging in the usual way took about 10 min., during which time much smoke and dust entered the building. The charging-opening in the roof, 10¼ by 7⅞ in.,



FIGS. 120 to 123.—Reverberatory hand-roasting furnace with level hearth.

[1] Von Der., Ropp, *Min. Sc. Press.*, 1902, LXXXV, 308, Selby, Cal.

[2] Hahn, *Tr. Inst. Min. Met.*, 1899–1900, VIII, 248, Monterey, Mexico.

[3] *Oest. Zt. Berg. Hüttenw.*, 1905, LIII, 4.

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lined with a cast-iron frame, carries a steeply inclined fixed steel plate, ⅛ in. thick, and a swinging plate suspended by trunnions in such a way that when the hopper is empty, the movable plate closes the space between the cast-iron lining and the fixed plate, and when it is being filled, the plate recedes and allows the charge to drop on to the hearth.

Temperature measurements in such a furnace, 58 ft. long, roasting leady matte, have been made by Clevenger.[1] They are shown in Fig. 124. The temperatures on side *A* were taken after moving the charge forward and during the highest heat; those on side *B* at the low heat prevailing when the charge is to be drawn.

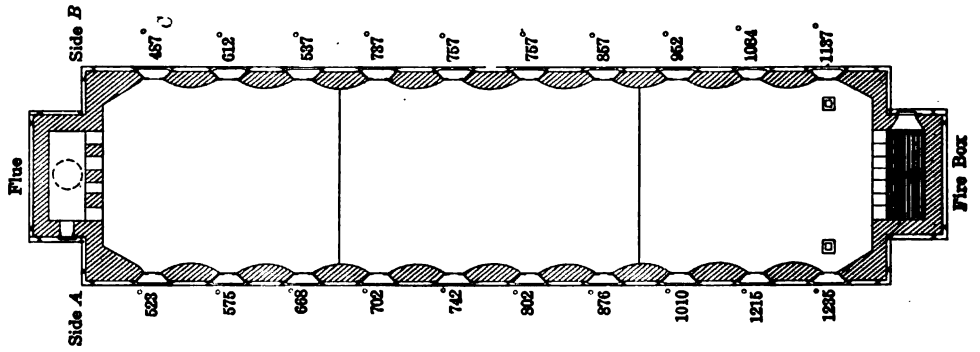

FIG. 124.—Temperature measurements in reverberatory hand-roasting furnace with level hearth.

The work done in this class of furnaces is shown in Table 42.

The amount of flue-dust formed varies with the size of the concentrate treated;[2] it is, however, small, and usually varies from 1 to 2 per cent. The fuel consumed is about 2 tons coal or 4 cords wood in 24 hr. The labor required varies with the tonnage; a fair figure is 2 men on a 12-hr. shift for 9 tons raw ore in a furnace with hearth 60 by 14 ft. If the furnace is 80 ft. long and puts through 12 tons in 24 hr., an additional man will be required on a shift.

If ore is to be slag roasted in a furnace with level hearth about 60 ft. long, this has near the firebridge a depression or sump less than 1 ft. deep in which the ore is melted. The consequence of this is that the ore in front of the third and fourth doors from the firebridge is pasty, and if the fire has not been carefully watched the heat may be excessive up to the fifth door. Not only does this interfere with a good roast, but it also renders the moving of the ore with the paddle a very arduous piece of work. To counteract this, it has been and often still is the custom with such furnaces to collect the ore from the third and fourth doors in a heap in front of the second door, and to melt it down into the sump, whence it is removed through the first door.

[1] *Met. Chem. Eng.*, 1913, XI, 447.

[2] *Eng. Min. J.*, 1910, LXXXIX, 449 (von Bernewitz); 857 (Hoke); XC, 112 (Wright).

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TABLE 42.—REVERBERATORY HAND-ROASTING FURNACE WITH LEVEL HEARTH

	I	II	III
Length of hearth	60′	66′	75′
Width of hearth	14′	16′	14′
Hearth area, square feet	840	1,056	1,050
Length of grate	8′	7′ 9″	8′
Width of grate	3′ 4″	2′ 6″	3′ 6″
Grate area, square feet	14.6	19.4	28
Ratio hearth to grate area	57½ : 1	54½ : 1	41 : 1
Space above firebridge, length and width	7′ 9″ × 2′ 2″	7′ 9″ × 2′ 2″	2′ 6″ × 1′
Space above fluebridge, length and width	No fluebridge	4′ 2″ × 8″	No fluebridge
Height of firebridge above hearth	14″	12″	20″
Height of roof above firebridge	18″	20″	12″
Height of fluebridge above hearth	.....	6″	.....
Height of roof above fluebridge	.....	15″	.....
Depth of grate below top of bridge	14″	15″	17″
Character of ore	⅔ Pyrite, ⅓ Galena	Matte Concentrates	Pyritic galena
Composition of ore	(*b*)	.....	(*a*)
Screen-size of ore (mesh)	2	12 and under	2
Depth of charge near fluebridge	3–4″	.....	5″
Ore stirred every (minutes)	30	20	40
Roasted ore drawn every (hours)	8	4	4
Time ore remains in furnace (hours)	32	24	24
Tons of raw ore in 24 hr	8.1	12	9
Pounds of ore roasted per square foot of hearth area	20	21.8	15.65
Character of roasted ore	Partly sintered	Pulverulent	Partly sintered
Per cent. sulphur in roasted ore	1.2(*c*)	2–5	3

(*a*) SiO_2 16–5, Fe 28–25, Pb 12–25, Cu 2–3, Zn 4–6, S 33, Ag 25–40 oz., Au 0.08–0.30 oz. (*b*) Pyrite: Fe 37, SiO_2 5, Cu 1, Zn 4–5; Galena: Pb 45–60, Zn 10–12. (*c*) Best roast.

A furnace of this description was in operation at Mine La Motte, Mo.,[1] where a galena concentrate was slag roasted. The hearth, 55 by 11½ ft. on the inside, was slightly inclined from flue to bridge, making the respective distances from hearth to horizontal roof 15 and 22 in. The top of the bridge-wall, 22½ in. wide, was 9 in. above the hearth and 13 in. below the roof. The grate, 10 ft. by 21 in., was 3 ft. 6 in. below the top of the bridge, this depth being necessary on account of the fuel used, which was wood. Of special interest is the construction of that part of the hearth where the ore was fused, and of the bridge. The former was built into a wrought-iron pan resting on brick pillars between which air circulated freely. It was formed by a full course of fire-brick, and was slightly concave. The air-cooling has proved very effective in preventing the corrosion of the brick. A similar result is produced by the air-flue passing through the bridge. When this was at first constructed in the usual way, it was found that the middle part of the bridge was apt to be eaten

[1] Private communication from J. T. Monell, May, 1891.

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through by the slagged ore. As a central wall divided the fireplace into two parts, the idea was conceived of closing the air-flue in the middle and erecting a small chimney on the roof, communicating with both parts of the air-flue. By this means a strong current of air could be passed through the flue. The experiment was a success, and was adopted.

The writer has used a water-jacket in fusing antimoniate of lead in a reverberatory furnace, and has found that it stopped all leakage at the bridge.

The Mine La Motte furnace was charged every 6 hr. with 2 tons of galena concentrate, to which some sand had been added as acid flux. The thickness of bed was 6 in. The galena ran from 40 to 70 per cent. Pb, and from nothing to 25 per cent. Fe, and was crushed to pass a 12-mesh sieve. The slagged ore retained from 4 to 6 per cent. S; four men worked on a 12-hr. shift, and 0.42 cord of wood was burned per ton of ore.

The furnace at Bonne Terre, Mo.,[1] was similar to the one at Mine La Motte. The roasting hearth was 40 ft. long by 11 ft. wide, the sinter hearth 11 ft. square and 8 in. deep. The grate was 7 ft. by 2 ft. 6 in., the firebridge 20 in. above the grate and 16 in. above the sinter hearth, and the roof 12 in. above the bridge. At the flue-end of the furnace was a bridge 8 in. high, and the roof was 16 in. above the fluebridge. The ore treated was a galena concentrate, 5 mm. and smaller, with Pb 72, Fe 4, Ca(Mg)O 5, S 15 per cent. It was charged to a depth of 4 in., stirred on the roasting hearth every ½ hr., on the sinter hearth every hour, and drawn every 6 hr. It remained 30 hr. in the furnace and retained 3.5 per cent. S. The furnace roasted 5 tons of raw ore in 24 hr. or 22.5 lb. per sq. ft. of hearth area.

81. Reverberatory Hand-roasting Furnace with Slagging Hearth or Fusebox.—This furnace, now abandoned, is represented in Figs. 125–130. The main improvement of the furnace, first erected at the Omaha and Grant Works, Denver, Colo., consists in carrying on the roasting on a hearth separate from that on which the slagging or fusing takes place, the ore being made to drop through a vertical flue, from 22 to 24 in. high, on to the slagging hearth. The hearth, as seen in Fig. 127, has the form of a reverberatory smelting furnace. In order to obtain the desirable sudden change from the high temperature in the slagging hearth to the low temperature in the roasting hearth, the flue-space above the latter is very much enlarged. Thus the flame on leaving the slagging hearth (Figs. 126 and 127) passes through a flue 5 ft. 6 in. by 1 ft. 4 in., which on entering the roasting hearth is suddenly enlarged to 17 ft. by 2 ft. 6 in. The sudden increase of area causes a correspondingly sudden decrease in temperature, and this produces the sudden change from pasty or fused to powdery ore. A detailed description of the furnace is not necessary, as the drawings can be understood without it. A few remarks, however, may be in place.

The roasting hearth is in four separate planes, divided by 3-in. offsets, which serve to keep the charges apart. The distance between roof and hearth is thus diminished by stages, leaving the former horizontal. This can also be

[1] Private communication from G. Setz, March, 1897.

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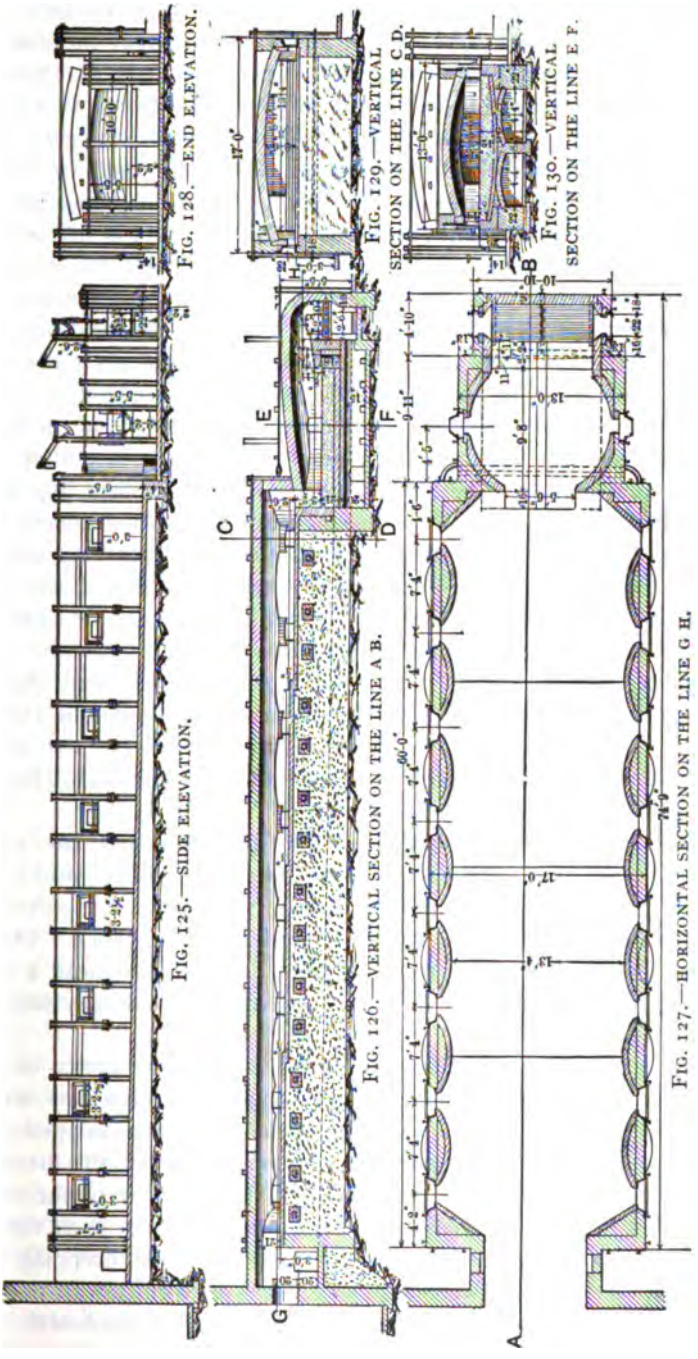
FIG. 125.—SIDE ELEVATION.
FIG. 128.—END ELEVATION.
FIG. 126.—VERTICAL SECTION ON THE LINE A B.
FIG. 129.—VERTICAL SECTION ON THE LINE C D.
FIG. 130.—VERTICAL SECTION ON THE LINE E F.
FIG. 127.—HORIZONTAL SECTION ON THE LINE G H.

FIGS. 125 to 130.—Reverberatory hand-roasting furnace with slagging-hearth or fuse-box.

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done with a single inclined hearth, which was advocated by Hodges,[1] because the offsets furnishing points of attack, lead to the injury of the hearth, and are not required to separate one charge from another. Only that part of the roof above the lowest roasting hearth is built of fire-brick; the rest is of red brick. In the end view (Fig. 128) are seen four openings for admitting air into the roasting hearth. The additional air required enters through the two doors next to the flue, which leads into the fuse-box, the door-lids being left slightly ajar. The working bottom of the fuse-box used to be (and is still sometimes) made of quartz sand seasoned with small amounts of slag which are added after the sand has been put into the furnace and heated until it becomes slightly sintered on the surface. This bottom is represented in the drawing. It has not proved as satisfactory as was expected, and has been generally replaced by a 9-in. fire-brick bottom, built slightly concave. The bottom rests on two arched roofs, and is thus cooled by air circulating below it. In the firebridge there is on one side of the air-space a heavy cast-iron bridge-plate to bear the longitudinal stress of the hearth. The parts of the furnace that wear out fastest are the flue leading from the fuse-box to the roasting hearth and the fuse-box itself; the former is patched during the run with raw clay mixed with some burned clay; to repair the latter the furnace has to shut down. Water-cooling of these parts has not been tried, as far as the writer is aware. The following analysis of a torn-out quartz bottom easily explains how the corrosion has taken place: SiO_2 43.6, FeO 31.5, MnO 1.0, Pb 6.5 per cent., Ag 6.1 and Au 0.5 oz. per ton. The cost of building a furnace in Pueblo or Denver, as shown in the drawings, used to be $3,000. The materials required are: Cast iron, 12,000 lb.; wrought iron, 4,000 lb.; sheet iron, 400 lb.; old rail buckstays, 10,000 lb.; red brick, 86,000; fire-brick, 15,000.

The tools required by each roaster-man are: Two paddles (blade 5 by 8 in. of $\frac{1}{8}$-in. iron, handle 10 or 12 ft. long of 1-in. iron); two rabbles (head 3 by 9 in. of $\frac{1}{8}$-in. iron, handle 12 ft. long of $\frac{3}{8}$-in. iron); one slice-bar ($1\frac{1}{2}$-in. iron rod flattened to a chisel-point 3 or 4 in. wide), and two door hooks. The front-man has two scoops for the coal, two slice-bars, three rabbles (head 4 by 9 in., of $\frac{1}{2}$-in. iron; handle 10 ft. long of $\frac{7}{8}$-in. iron), and the necessary slag-pots to receive the slagged ore.

The mode of working a furnace with a fuse-box used to be pretty uniform at the different smelting works. The ore crushed to pass a 2-mesh sieve was sometimes dried on the roof of the furnace, but generally it was dropped directly, without drying, through the hopper into the coolest part of the furnace, the weight of the charge varying from 2400 to 3300 lb. according to the thickness of bed the ore can bear. Sometimes the ore was shoveled on to the hearth through the last two doors, but this is permissible only when dropping through a hopper is impracticable. The charge was spread uniformly with paddle and rabble to a thickness of about 4 in. over the highest point of the hearth. If this is not separated by steps, but merely inclined, the charge is so spread that it shall lie in front of the first two doors. There it remained until slagged ore

[1] *Eng. Min. J.*, 1885, XL, 286.

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was drawn from the fuse-box, when it was moved down the furnace to its second place on the next hearth, or in front of the next two doors. During its journey to the fuse-box it was not only turned over with the paddle while being moved, but raked with the rabble once, twice, or three times, according to the interval of time between the movings, *e.g.*, every 1½ hr. with 3-hr. movings. Before the charge was transferred to the fuse-box the latter received some siliceous ore to protect the bottom. After dropping the charge the fire was urged. The liquefying begins at the surface, and much rabbling is required to bring the unfused parts from the bottom to the top. This was done at intervals of ½ hr. for 15 min. at a time at the beginning, later on for 10 min., and toward the end for 5. Sometimes the fusing ore was rabbled only once, 1½ hr. after dropping from the roasting hearth. When the charge was fused, it was drawn into slag-pots. This may be done in two ways, either all at once or in three installments. The former method was used with charges containing 10 per cent. Pb, the latter with those running high, 50 and 60 per cent. Pb, for if they remain any length of time in contact with the acid hearth material they will take up SiO_2 and corrode the hearth unnecessarily. As soon as the front-man had drawn the slagged ore from the fuse-box, the roaster-men dropped the next charge and began to transfer the other charges downward, until the hearth near the flue was emptied and ready for a new charge. When the fused ore had become cold, it was dumped from the slag-pots, broken up, and transferred to the feed floor of the blast furnace. Its composition of course must vary greatly. When made from mixed sulphide ores, it will average: SiO_2 30 to 40, Fe 21 to 25, Zn 9 to 10, Pb 10 to 15, S about 2 per cent.

A furnace like the one shown in the drawings roasted and fused in 24 hr. from six to eight charges (varying in weight from 2400 to 3300 lb.), consumed from 3 to 4 tons of bituminous coal, half lump and half pea, and required three men in a 12-hr. shift—one front-man, who attended to the fuse-box and firing, and two roaster-men, who did the work on the roaster hearth. If the ore is not fused, two men are sufficient to do the work. The cost of roasting and fusing with coal at $1.75 per ton, and labor $2.25 for 12 hr., was over $2 per ton.

In making up sulphide ore-beds for fusing furnaces two kinds of ore have to be considered: galena concentrates free from or low in silver, assaying 50 to 60 per cent. Pb, and mixed argentiferous sulphides containing 10 per cent. Pb and less. With the former class of ore Cramer von Clausbruch[1] stated that at the Altenau smelting and refining works (Harz Mountains) he obtained the best results in treating his galena ores, if the charge contained 15 per cent. SiO_2 and from 55 to 60 per cent. Pb. If there was more SiO_2, a base had to be added to effect a complete slagging at a reasonably low temperature; if there was less, some $PbSO_4$ remained undecomposed. He notes the interesting fact that, if the roasted ore is not completely slagged, but retains parts of sulphides and sulphates that have been only agglomerated, the silver and copper will be concentrated in the agglomerated part. One hundred parts of his charge gave 85 per cent. of thoroughly slagged ore, 10 per cent. of a mixture of slagged and

[1] *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1883, xxxi, p. 26; *Eng. Min. J.*, 1883, xxxv, 161.

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agglomerated ore, and from 2 to 3 per cent. of unroasted agglomerated galena, the loss in roasting varying from 2 to 3 per cent. The slagged part of the charge contained one-half of the Ag and only a trace of Cu, while the other half of the Ag and all the Cu were concentrated in the rest of the charge.

At Přibram[1] the beds of galena concentrate contained SiO_2 20, Pb 35 per cent., and Ag 76 oz. per ton; at Mechernich,[2] SiO_2 22, Pb 58 per cent., Ag 4 oz. per ton. Analyses of roasted ore are given in Table 43.

With mixed ore the principal base to combine with the SiO_2 of the sulphide ore will be Fe, and the one next in importance Pb. Charges were made up so that they might be readily fusible and sufficiently acid not to corrode the bottom and side walls of the fuse-box. They should contain from 25 to 32 per cent. SiO_2, and the iron calculated as Fe be made to equal the SiO_2.

Sometimes, however, proportions such as SiO_2 10 to 15, Fe 20 to 28, Pb 20 to 28 per cent. were chosen, owing to the varying amounts of lead and iron sulphide ore on hand, from which the charges had to be made up. It was not common to add lime to a charge containing lead.

82. Reverberatory Hand-roasting Furnace with Sinter Hearth.—This furnace, which is still found here and there, resembles very much the one with a slagging hearth, the main difference being that, on account of the low temperature required for sintering, the sinter hearth has the same width as the roasting hearth and is separated from the latter by a step, say 8 in. high, and that the roof of the sinter hearth has less pitch.

The main dimensions of such a furnace are: Roasting hearth, 74 by 14 ft.; sinter hearth, 10 ft. 4 in. by 14 ft.; grate, 9 ft. 9 in. by 2 ft. 10 in.; firebridge 9 ft. 9 in. by 1 ft. 6 in.; height of firebridge above grate, 11 in., above sinter hearth 11 in.; height of flue above firebridge, 12 in. at sides and 18 in. at center; height of step separating sinter and roasting hearth, 8 in.; height of flue leading from sinter to roasting hearth, the same as that of roasting hearth, viz., 17 in. at sides and 25½ in. at center. An ore charge, crushed through a 2-mesh sieve and consisting of pyrite with galena and blende, contains, SiO_2 10 to 25, Fe 20 to 30, Pb 2 to 15, Cu 1 to 3, Zn 2 to 10, S 28 to 44 per cent.; Ag 15 to 35 oz., and Au up to 1 oz. per ton. At the Eilers plant, Pueblo, Colo., a charge usually ran SiO_2 15, Pb 15, Fe 35, Zn 8, S 5 per cent.[3] A charge remains 30 hr. in the furnace, is stirred on the roasting hearth every ¾ hr., on the sinter hearth every ½ hr., drawn every 3 hr., and retains 2 to 3 per cent. S. Sintered ore, while it is being drawn from the furnace into a slag-pot, is usually pounded down with an iron disk, 1 ft. in diameter, attached to a handle.

The furnace puts through in 24 hr. 12 tons of raw ore, requires three men on a shift, and consumes about 3 tons of coal.

83. Products of Reverberatory Hand-roasting Furnaces.—The two products obtained by roasting are roasted ore and flue-dust. The compositions of some

[1] Zdráhal, *Oester. Jahrb.*, 1880, XXXVIII, p. 10.
[2] *Berg. Hüttenm. Z.*, 1875, XXXIV, 129.
[3] Gilbert, F. C., May, 1904.

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TABLE 43.—ANALYSES OF ROASTED ORES FROM REVERBERATORY HAND-ROASTING FURNACES

	Rodna, Transylvania		Mechernich, Prussia		Freiberg, Saxony	Hall Valley, Colo.	Mine La Motte, Mo.	Omaha & Grant, Denver, Colo.		Globe Works, Denver, Colo.				Leadville, Colo.		
	Raw ore	Sintered ore	Raw ore	Slag-roasted ore	Slag-roasted ore	Roasted ore	Sintered ore	Slag-roasted ore		Sintered ore				Fused pyritic ore		Roasted ore
Pb	47.29	54.27	60.40	62.08	.....	.....	75.95	13–15	13.15	20.5	21.8	15.0	27.0	.....	.....	.....
PbO	.....	.....	.....	.....	22.0	42.04	.....	.....	.....	.....	.....	.....	.....	.....	.....	27.06
Ag	0.059	0.061	0.0105	.....	0.13	Ag_2O, 0.21	.....	.....	.....	28.0	91.5	57.0	75.0	28.0	30.0	42.75
Au	0.0001	0.0001	.....	.....	.....	.....	Ni-Co, 1.11	.....	.....	0.26	0.24	0.51	0.24	0.13	0.12	.....
Cu	Trace	0.02	0.17	0.14	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Cu_2O	.....	.....	.....	.....	0.3	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
CuO	.....	.....	.....	.....	.....	1.71	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.68
As	0.34	0.030	NiO.10	N. d.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
As_2O_5	.....	.....	.....	.....	1.1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	Mn_3O_4
Sb	0.02	0.027	0.07	0.08	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.23
Fe	20.36	24.06	0.80	0.56	.....	.....	.....	25.3	22.7	30.8	21.8	32.6	29.2	.....	.....	.....
FeO	.....	.....	.....	.....	.....	3.59	4.5	.....	.....	.....	.....	.....	.....	53.1	74.8	.....
Fe_2O_3	.....	.....	.....	.....	33.3	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	32.69
Zn	0.67	0.87	0.15	N. d.	16.0	.....	.....	10.5	9.5	4.6	7.3	4.0	4.1	.....	.....	.....
ZnO	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	30.82
Al_2O_3	0.11	0.23	3.60	4.24	1.8	8.11	.....	.....	.....	.....	.....	.....	.....	.....	.....	2.31
CaO	Trace	Trace	0.88	1.28	2.0	0.42	0.76	.....	.....	.....	.....	.....	.....	.....	.....	1.36
MgO	Trace	Trace	.....	.....	0.5	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
BaO	.....	.....	.....	.....	.....	12.05	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
SiO_2	0.49	0.80	22.05	27.77	17.4	22.71	7.21	34.7	35.7	15.0	22.0	18.4	10.4	4.7	4.0	4.64
CO_2	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
SO_3	.....	2.25	.....	.....	Trace	6.31	.....	.....	.....	.....	.....	.....	.....	.....	.....	4.05
S	29.86	2.72	9.72	0.60	3.6	2.94	7.41	.....	.....	3.8	4.0	2.8	2.1	.....	6.0	1.00
O	.....	13.41	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Reference	(a)	(a)	(b)	(b)	(c)	(d)	(e)	(f)	(f)	(g)	(g)	(g)	(g)	(h)	(h)	(i)

(a) *Oesterr. Jahrb.*, 1881, XXIX, 27. (b) *Berg. Hüttenm. Z.*, 1875, XXXIV, 129. (c) *Oesterr. Jahrb.*, 1866, XVI, 397. (d) *Tr. A. I. M. E.*, 1876–77, V, 568. (e) *School Mines Quart.*, 1887–88, IX, 216. (f) Livingstone, private communication, August, 1896. (g) Iles, private communication, August, 1896. (h) Private notes. (i) Dewey, Bulletin 42, U. S. National Museum, 1891, p. 48.

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ores running high in lead that have been agglomerated or completely slagged is given in Table 43.

An analysis of crystals of slag-roasted galena ore rich in zinc gave, according to Heberdey,[1] SiO_2 16.62, PbO 61.50, ZnO 18.26, FeO 1.69, CaO trace, MgO 1.99 per cent.; sp. gr. 5.214.

The amount of flue-dust carried off with the gases is about 2 per cent., and is practically all collected in dust-chambers; of the metal volatilized in fusing very little if any is recovered, unless a Cotterell precipitator is provided. Flue-dust from furnaces, if the ore is simply roasted, has a brownish color; if slagging is carried on, it is gray from volatilized lead and zinc. This gray dust, if it contains not less than from 1 to 2 per cent. Zn has, when moistened, the property of solidifying to a hard mass. This is not the case with brown flue-dust. The solidifying is probably caused by the anhydrous zinc sulphate becoming hydrated. Its binding property is strong enough to allow the mixing in of 25 per cent. of non-binding material, when it still will form a hard brick when molded. In fact, the gray dust, when moistened, forms a pasty mass requiring a stiffening ingredient to permit its being molded at all. The compositions of some samples of flue-dust are given in Table 44.

The large quantity of gold shown in some of the analyses is probably due to gold-bearing pyrite concentrates from gold mills used as iron flux, and to telluride gold ores.

84. Blast Roasting, General.[2]—The aim of blast roasting is to roast and agglomerate finely divided metallic sulphide by means of forced draft. In order to accomplish this, it is necessary that the particles, brought to ignition temperature, be separated from one another and be in contact with fluxes so that the grains of sulphide may be completely oxidized, and that the heat generated thereby may be sufficient to cause the oxides formed to combine with the fluxes and form fusible mixtures or compounds which, being pasty at the prevailing temperature, cohere to one another and form a lump of more or less scorified material.

The advantages of blast roasting over the usual oxidizing roasting have been so great that the former process has almost wholly replaced the latter in lead smelteries. Some of the advantages are: Oxidation and elimination of sulphur, combined with agglomeration of ore accompanied by decrease in volatilization of metal, and reduction of cost of plant and of treatment. With regard to subsequent smelting in the blast furnace: Increase of smelting power to as much as 100 per cent. owing to absence of fines, intimate mixture of charge-components in and porosity of blast roasted cake or sinter, decrease in requirement of coke and of pressure of blast, and, with the last, decrease of loss of metal by dusting and volatilization.

[1] Kroupa, *Oesterr. Zt. Berg. Hüttenw.*, 1893, XLI, 138.

[2] Hofman, "General Metallurgy," 1918, 411; *Tr. A. I. M. E.*, 1910, XLI, 739.
Ingalls, *op. cit.*, 1906, XXXVII, 627.
Guillet, *Rev. Mét.* 1911, VIII, 584.

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TABLE 44.—ANALYSES OF FLUE-DUST FROM REVERBERATORY HAND-ROASTING FURNACES

	Friedrichshütte, Silesia (*a*)	Freiberg, Saxony (*b*)			Přibram, Bohemia (*c*)	Omaha & Grant Works, Denver, Col. (*d*)	Globe Works, Denver, Colo. (*e*)		Globe Works, Denver, Colo. (*f*)	
Pb	.....	26.27	21.37	16.27	.....	.....	18.70	12.60	5–10	19–40
PbO	62.80	.....	.....	.....	45.50	.....	.....	.....	.....	.....
$PbSO_4$	.....	.....	.....	.....	.....	57.23	.....	.....	.....	.....
Ag, oz.	23.33	.....	.....	.....	46.66	.....	37.70	46.60	23–28	20–37
Au, oz.	.....	.....	.....	.....	.....	.....	0.15	0.26	0.50–1.50	0.10–0.75
Cu	.....	.....	.....	0.02	Trace	.....	2.70	2.70	.....	.....
Cl	.....	.....	.....	.....	Trace	.....	.....	.....	.....	.....
Sb_2O_3	.....	.....	.....	.....	1.35	.....	.....	.....	.....	.....
As	.....	7.56	37.5	46.41	.....	.....	0.30	0.30	.....	.....
As_2O_3	.....	.....	.....	.....	1.06	15.86	.....	.....	.....	.....
Zn	.....	19.10	2.11	0.45	.....	.....	6.30	6.10	None	7–8
ZnO	3.20	.....	.....	.....	2.08	1.50	.....	.....	.....	.....
Fe	.....	.....	.....	.....	.....	.....	13.80	.....	.....	.....
FeO	.....	.....	.....	.....	.....	1.30	.....	.....	.....	.....
Fe_2O_3	4.44	.....	4.36	1.63	.....	.....	.....	.....	.....	.....
Fe_3O_4	.....	.....	.....	.....	7.20	.....	.....	.....	.....	.....
MnO	.....	.....	.....	.....	1.14	.....	.....	.....	.....	.....
Al_2O_3	.....	.....	.....	.....	0.53	.....	.....	.....	.....	.....
CaO	1.96	0.80	0.61	0.43	0.45	.....	.....	.....	.....	.....
MnO	.....	0.50	0.25	0.15	.....	.....	.....	.....	.....	.....
SiO_2	.....	8.90	4.81	2.53	(=10.0)	.....	16.10	.....	.....	.....
Insol.	.....	.....	.....	.....	14.20	.....	.....	.....	.....	.....
SO_3	25.80	28.14	8.23	7.95	.....	.....	.....	18.20	.....	.....
H_2SO_4	.....	.....	.....	.....	23.28	17.13	.....	.....	.....	.....
S	.....	.....	1.34	0.60	Trace	.....	8.20	8.60	.....	.....
C	.....	.....	3.40	3.75	2.50	.....	.....	.....	.....	.....
H_2O	1.50	.....	1.46	3.27	.....	.....	(*g*)	(*h*)	(*i*)	(*j*)

(*a*) Kosmann, *Zt. Berg. Hütten. Sal. Wes. i. Pr.*, 1883, XXXI, 227. (*b*) Hering, "Die Verdichtung des Hüttenrauches," Stuttgart, 1888, p. 34. (*c*) Mann, *Oester. Jahrb.*, 1890, XXXIX, 15. (*d*) Livingstone, private notes, 1896. (*e* and *f*) Iles, private notes, 1896. (*g*) Average of 10 years. (*h*) From fusing furnaces. (*i*) From sintering furnaces. (*j*) From roasting furnaces.

The earliest record of the practical application of the principle of forced roasting is that of Roswag,[1] who describes operations carried on in Spain in a spherical kettle. An earlier instance, different as to details, is that by Wassiliadi,[2] who discusses some work carried on in Asia Minor in 1895.

Blast roasting became important only with the advent of the Huntington-Heberlein process, in which a galena concentrate mixed with crushed limestone is rough-roasted in a reverberatory furnace, moistened, and then blown in a spherical kettle. For this process the term *Lime Roasting*[3] was used for some time. In the development of the practice of blast roasting it was found that

[1] Frémy, "Encyclopédie Chimique," Dunod, Paris, 1884, section 1, vol. 5, p. 243; *Eng. Min. J.*, 1912, XCII, 750.

[2] *Tr. Inst. Min. Met.*, 1912–13, XXII, 702.

[3] Ingalls, *Eng. Min. J.*, 1905, LXXX, 402.

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lime was not of vital importance; the term *Pot Roasting*[1] was therefore an improvement. With the advent of the Dwight-Lloyd roast-sintering machines the kettle or pot ceased to be an essential factor; the broad term *Blast Roasting*[2] has replaced the two preceding designations, although they are still used occasionally. Terms like "bessemer roasting" and "blast and down-draft sintering"[3] have been suggested, but not adopted.

The second step in blast roasting was that taken with the Carmichael-Bradford process, in which a moistened mixture of raw galena concentrate and dehydrated gypsum is blown in a pot.

The third modification was that of the Savelsberg process, in which crushed raw galena is mixed with limestone, moistened, and blown in a spherical kettle.

The high calorific power of galena concentrate diminished by a rough-roast in the Huntington-Heberlein process, is reduced in the Bradford-Carmichael and the Savelsberg processes by large additions of gypsum and limestone.

These three processes were proposed for the treatment of sulphide lead ore. They differ in the preparation of the charge, but are very much alike in the manner of blowing, in that they use a deep spherical kettle with blast-inlet at the bottom, charge the kettle, blow the content, and discharge the cake that has been formed. They use *Up-draft* and are *Intermittent*.

The fourth and last step in blast roasting is that taken with the Dwight-Lloyd process, which at the start was not confined to lead ores and hence not to certain fluxes or diluents. The apparatus differs radically from its predecessors in that a thin layer of ore mixture on a traveling grate is made to pass over a stationary suction-box where, after ignition on the surface, down-draft causes the roast to progress downward, and the traveling grate discharges the blast roasted ore in the form of cakes into a receiver. The leading features of the process are therefore a thin layer of ore, *Down-draft*, and a *Continuous* operation.

The great success of blast roasting lead ores has been the cause of extending the operation to other sulphides, *e.g.*, to concentrates of sulphide copper and copper-nickel ores, and to matte, in preparing them for blast furnace treatment. Arsenides, such as speiss, have been treated successfully, as have oxides of iron and flue-dust which, however, require the addition of heat-producing material in the form of sulphide or carbonaceous fuel. With oxides, sintering is the main object and desulphurization only incidental.

85. Theory of Blast Roasting.—There have been published many speculations[4] about the reactions that might take place in blast roasting, especially in regard to possible interactions of PbS, $PbSO_4$, PbO, CaO and $CaSO_4$, and the

[1] Austin, *Min. Sc. Press*, 1906, XCIII, 511.
[2] Dwight, *Eng. Min. J.*, 1908, LXXXV, 649.
[3] *Met. Chem. Eng.*, 1910, VIII, 137, 222.
[4] Summaries: Prost., *Rev. Un. Min.*, 1907, XVIII, 303.
Kroupa, *Oesterr. Zt. Berg. Hüttenw.*, 1908, LVI, 276.
Editor, *Eng. Min. J.*, 1909, LXXXVII, 613.
Gmelin-Kraut, "Handbuch der anorganischen Chemie," 1912, vol. 4, part 2, p. 33.
Richter and Bannister, *see below*.

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rôle and effect of SiO_2 and of bases usually present in a lead-charge. Valuable research has been carried on to give precise knowledge of the chemical reactions that may take place.[1] The most reasonable explanation of blast roasting as a whole is that formulated by Richter, who says that blast roasting is nothing more or less than an enforced ordinary roast similar to that taking place in the hand reverberatory furnace, in which the reactions taking place with galena may be expressed by

$$PbS+3O=PbO+SO_2+99{,}866 \text{ cal.}$$
$$SO_2+O=SO_3+22{,}640 \text{ cal.}$$
$$PbS+4SO_3=PbSO_4+4SO_2+104{,}940 \text{ cal.}$$
$$PbO+SO_3=PbSO_4+73{,}000 \text{ cal.}$$
$$2PbO+SiO_2=Pb_2SiO_4+x \text{ cal.}$$
$$2PbSO_4+SiO_2=Pb_2SiO_4+2SO_3 \text{ (or } SO_2+O)$$

The large volume of air, blown or drawn into the ore charge, surrounds the single particles of sulphide, drives off the SO_2 as soon as formed, and thus counteracts to some extent the formation of sulphate; at the same time it causes the oxidation to proceed at such a speed that the heat generated causes the resulting oxide to form a sintered mass with the accompanying gangue or the added flux.

With a faulty charge there may occur with exclusion of air: $PbS+3PbSO_4=4PbO+4SO_2-187{,}060$ cal.; or through lack of SiO_2 metallic lead may be set free by the endothermic reactions between PbS and $PbSO_4$ or PbO; in the presence of air at an elevated temperature there may take place: $PbS+PbSO_4+2O=2PbO+2SO_2-4{,}220$ cal.

The components CaO and $CaSO_4$ do not appear to have any direct chemical action upon PbS and PbO, as seen by the following experimental facts: $PbS+CaO+700°$ C. $=PbO+CaS$ does not take place (Richter); $PbS+CaSO_4+$

[1] Clark, *Eng. Min. J.*, 1904, LXXVIII, 630, 708.
Borchers, *Metallurgie*, 1905, II, 1.
Savelsberg, *op. cit.*, 1905, II, 5.
Hutchings, *Eng. Min. J.*, 1905, LXXX, 726; 1907, LXXXIII, 201.
Guillemain, *Metallurgie*, 1905, II, 433; *Eng. Min. J.*, 1906, LXXXI, 470; *Metallurgie*, 1910, VII, 599.
Ingalls, *Eng. Min. J.*, 1905, LXXX, 1067.
Doeltz, *Metallurgie*, 1905, II, 460; *Eng. Min. J.*, 1906, LXXXI, 175.
Weiller, *Chem. Z.*, 1908, XXXII, 348.
Tandler, *op. cit.*, 1908, XXXII, 437.
Hofman-Mostowitsch, *Tr. A. I. M. E.*, 1909, XXXIX, 628, XL, 807.
Richter, Paul, "Beiträge zur Theorie des Huntington-Heberlein Processes und der ihm Verwandten Blaseverfahren," Dissertation, Dresden, 1909.
Hofman, *Eng. Min. J.*, 1911, XCI, 48.
Schütz, *Metallurgie*, 1911, VIII, 228.
Bannister, *Tr. Inst. Min. Met.*, 1911–12, XXI, 346.
Hahn, *Eng. Min. J.*, 1911, XCI, 1284 (laboratory experimental apparatus).
Pulsifer, *op. cit.*, 1913, XCVI, 308 (laboratory experimental apparatus).

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$CO_2+850°$ C.$=PbS+CaS$ does not take place (Doeltz); $PbO+CaSO_4+$ heat$=CaPbO_3+SO_2$ does not take place.[1]

The observations of Percy and Hutchings as well as the curves of Bannister (Fig. 33) show that in using CaO as flux there takes place an exothermic reaction the character of which is not known. Practical experience also teaches that with lead ores lime has a favorable influence upon desulphurization, but the cause of it has not been ascertained.

To what extent $CaSO_4$ is formed in a blast roast is not known. Bannister in using CaO in his mixture obtained large amounts of $CaSO_4$; but in using $CaCO_3$ on a large scale it is believed that, judging by the experience in roasting sulphide copper concentrates in a McDougall furnace,[2] the $CaCO_3$ will be found unchanged or only little altered in the rough-roasted charge. In blast roasting $CaSO_4$ may be formed. This is decomposed by heat alone at 1200° C.[3] The decomposition by SiO_2, as shown by $CaSO_4+SiO_2=CaSiO_3+SO_2+O$, begins at 1000° C. and is finished at 1250°.[4] In the presence of galena the decomposition, $6CaSO_4+6SiO_2+2PbS=6CaSiO_3+8SO_2+2PbO$, begins below 800° C., is complete at 1000–1100°, and with $BaSO_4$ at 1100–1200°,[5] the SO_3 set free is completely dissociated at 1000°[6] into SO_2 and O; in a cooler part of the charge SO_2 and O may recombine, and the SO_3 formed have an oxidizing effect. Lastly $CaSO_4+2Fe_2O_3=CaO.2Fe_2O_3+SO_2+O$ begins at 1100° C. and is finished at 1250° C.[7]

86. Principles Governing Make-up of Charges.[8]—In the preparation of a charge for blast roasting, be it up- or down-draft, certain requirements have to be fulfilled, if there is to be a satisfactory elimination of S, and if the powdery mixture is to be converted into a more or less sintered porous cake.

The heat furnished by oxidation must be sufficient for agglomeration and not in any great excess over the amount required. Comparing the quantities of heat liberated by oxidation per atom or 16 g. O of the usual charge-components gives the following:

$(Zn,O)=84{,}800$ cal.;
$(Fe,O)=65{,}700$ cal.;
$(Ni,O)=61{,}500$ cal.;
$\frac{1}{3}(Sb_2,O_3)=55{,}630$ cal.;
$\frac{1}{3}(As_2,O_3)=52{,}130$ cal.;
$(Pb,O)=50{,}800$ cal.;
$\frac{1}{2}(C,O_2)=48{,}600$ cal.;
$(Cu_2,O)=43{,}800$ cal.;
$\frac{1}{2}(S,O_2)=34{,}630$ cal.;
$\frac{1}{3}(S,O_3)=30{,}630$ cal.;
$(C,O)=29{,}160$ cal.

[1] Hofman-Mostowitsch, *loc. cit.*
[2] Hofman, "Metallurgy of Copper," 1918, 104.
[3] Hofman-Wanjukow, *Tr. A. I. M. E.*, 1912, XLIII, 523.
[4] Hofman-Mostowitsch, *op. cit.*, 1908, XXXIX, 645.
[5] Schütz, *Metallurgie*, 1911, VIII, 228.
[6] Hofman, "General Metallurgy," 1918, 405.
[7] Hofman-Mostowitsch, *loc. cit.*
[8] Hofman, *Eng. Min. J.*, 1911, XCI, 48.
Pulsifer, *Met. Chem. Eng.*, 1912, X, 153, 207.
Klugh, *Iron Trade Review*, 1915, LVII, 835.

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This shows that the oxidation of metal evolves a much larger amount of heat than does that of S. A low-grade copper matte, or one that is rich in Fe, will develop, with the same velocity of reaction, a higher temperature than a high-grade which runs low in Fe. The same is the case with a lead matte when compared with a galena concentrate. The charge for a material developing much heat has to be made up differently from one which develops little.

The speed with which the oxidation takes place is another factor. If it is too great, sintering will take place before roasting is finished, with the result that the charge becomes tight and does not allow air to pass freely, and that sintered parts enclose raw sulphide.

With regard to lead charges, it is observed by Mostowitsch[1] that a bi-silicate mixture, and one of lower degree of silication, runs hot and that the desulphurization is good (2 per cent. S and less); but that there is left unagglomerated a large part of the charge; that there is set free metallic Pb from the action of $PbSO_4$ or PbO upon PbS; and that there is present uncombined PbO in the product. The globules or veinlets of Pb toughen the cake, and the PbO makes dust when the cake is broken or handled. With an acid mixture, bi-silicate or higher, sintering begins before roasting has progressed satisfactorily, hence desulphurization is bad (5 per cent. S and higher). The cake breaks readily and makes no dust, but retains much sulphide-S.

There are several ways of correcting the heating effect of an excess of sulphide. One is to add an extra amount of water to the charge, *e.g.*, 15 as against the usual 5 per cent. The effect of water in the charge is three-fold: It counteracts dusting; it keeps down the temperature on account of the heat absorbed by its evaporation; it keeps the charge open owing to the spaces left unfilled when it has been expelled. Thus with ore ½ in. and smaller, an addition of 5 per cent. water may be sufficient, whereas a fine or flotation concentrate may require as much as 12 per cent. water if it is to work satisfactorily.

A second method is to rough-roast the ore, as is the case with the Huntington-Heberlein process, in order to convert some sulphide into sulphate and oxide. This not only diminishes the calorific power of the ore, but adds some oxide which becomes a diluent and acts as if a corresponding amount of extraneous matter had been added to the charge.

The addition of a diluent to the raw sulphide to reduce the calorific power, as is the case in the Bradford-Carmichael and Savelsberg processes, forms a remedy which quickly reaches an economic limit with regard to the subsequent smelting in the blast furnace. If the specific heat of the diluent is high, it will be more efficient than if it is low.

Retarding the process by the use of air weakened in its oxidizing power by the admixture of waste fuel-gases has been suggested by Savelsberg, but has not been tried.

The statement of Richter[2] that the character of the gangue has little influence

[1] *Tr. A. I. M. E.*, 1916, LV, 744.

[2] *Loc. cit.*

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in blast roasting is based upon roasting experiments made in an assay-muffle which showed that oxidation was hastened about equally by the 15 diluents he tried, namely, limestone, iron oxide, gypsum, caustic lime, dolomite, barite, witherite, baryta, cerussite, anglesite, lead- and copper-blast furnace slag, ground brick, silicious gangue, and blue billy (iron ore). The experiments of Bannister[1] with CaO, SiO_2, PbO+CaO, $CaCO_3$, $CaSO_4$, MgO, Fe_2O_3, H_2CaO_2, tell another story. As far as the mechanical effect of holding apart the sulphide particles and thereby favoring the speed of a roast is concerned, the statement of Richter holds true; but the thermal and chemical behaviors of the additions to a charge have to be well considered. It is essential for a blast roast that complete oxidation of the sulphide be accompanied or closely followed by sintering or fusing. The latter takes place at a temperature slightly higher than the former; with silicate mixtures the two curves usually run parallel. The specific heat of the gangue or flux[2] must be of such a quantity as to absorb any excess heat generated by the forced roast; it ought to act as a thermal balance-wheel which equalizes the temperature and holds it at a point at which roasting and sintering can proceed at the desired rate and in the required degree. If the specific heat is too low, the temperature rises too quickly and the charge fuses before it is sufficiently desulphurized; if too high, too much heat is absorbed, and the temperature of the sulphide is kept too low for a perfect roast and especially for the desired sintering with the result that part of the charge is imperfectly roasted and remains pulverulent. The following substances are arranged approximately in the order of their specific heats,[3] the first having the lowest value: lead oxide, cerussite, anglesite, barite, copper oxide, witherite, manganese oxide, iron oxide, iron silicate, calcium sulphate, basic copper carbonate, alumina, silicate low in iron, horn blende, limestone, sandstone, ground brick, dolomite, clay, copper- and lead-blast furnace slag, and gypsum.

As regards the chemical effect of the flux it is essential that the mixture have a composition which sinters at a low temperature, forming silicate or ferrite or both. For example, with lead ores the composition is usually one which will result in a singulo-silicate of low formation temperature. If the percentage of metallic sulphide is too low, raw sulphide or carbon will have to be added to the charge. A charge rich in Pb may contain from 9 to 12 per cent. S, one that is poor in Pb as much as 20 per cent.

The size of grain of the ore will be the same as that which has been found to be best for an ordinary roast, *e.g.*, for galena about 8-mesh. Difficulties arise in the treatment of flotation concentrates. These have been overcome in part by pugging with coarse concentrates in quantities to be found by experiment. The grain of the flux ought to be coarser than that of the ore; with a galena charge the size is about 4-mesh. With mixed ores the largest permissible

[1] *Loc. cit.*

[2] Mostowitsch, private communication, 1910.
Guillemain, *Metallurgie*, 1910, VII, 599.

[3] Landolt-Bernstein-Roth, "Tabellen," Springer, Berlin, 1912, 754.

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size is ½-in. Further, it is desirable that the grains be angular and not rounded in order that they may interlock, resist the pressure of the blast, and keep open the spaces necessary to furnish free access to the air. The flux will therefore be crushed with rolls and not with ball mills or similar apparatus which furnish rounded grains. Fine concentrates will have to be worked with coarse material, and the mixture well wetted.

87. Management of Furnaces, General.—With all blast roasting work there have to be considered the kindling of charge, the regulation of blast and of temperature, and the withdrawal of gases.

In an up-draft kettle a charge can be ignited by means of a thin bed of hot rough-roasted ore delivered from the reverberatory furnace to the kettle or to a layer of inert matter which covers the grate to protect it from the hot kindling-charge. Upon starting the blast, the hot rough-roasted ore will burn freely, and the heat generated will penetrate into the moistened ore-mixture above and ignite it. If pre-roasting is not practised, some carbonaceous fuel, usually half-burnt coal, will be charged on to the protecting layer (silicious ore, limestone); the coal may or may not be followed by easily burning pyritic ore, and this by moistened charge. The result of the roast is governed in part by a correct rate of kindling. If the temperature at the start is too high, the bottom of the charge will clinker and be less permeable to gases than it ought to be, and the charge will be imperfectly desulphurized. If the temperature is too low, the progress of the blast roast will be slow, and probably imperfect. Practical experience has shown that with a rich galena concentrate, kindling with hot pre-roasted ore does not work as satisfactorily as with glowing coal.

In a down-draft apparatus the flame playing upon the surface of the charge has to be regulated with care, as otherwise the same phenomena will occur as in an up-draft, only to a smaller degree.

The blast in an up-draft kettle is small in volume and low in pressure at the start, but both are increased later on. The pressure will rise on account of the resistance offered by the clinkered cake to the ascent of the air. At some works the volume of air is diminished toward the end of a blow, as less sulphide has to be oxidized than at the beginning, and as less dust is removed from the surface of the charge. At other works the volume of air is increased to a maximum after the charge has passed the kindling stage, and is held there to the end. In principle, the first mode of operating is more correct; in practice each case has to be studied separately.

In down-draft work, with its shallow bed of ore, there will be little or no variation in the volume of air drawn through the apparatus.

One of the essential points for a satisfactory oxidizing roast is the removal of sulphurous gas as soon as formed. As with a deep charge this is more difficult than with a shallow, special attention has to be given to this point with up-draft kettles, while with down-draft apparatus the suction which draws in air also removes the gases. There is danger with down-draft that S from pyritic ore may collect in fan and flue, and cause trouble, which is not the case with updraft.

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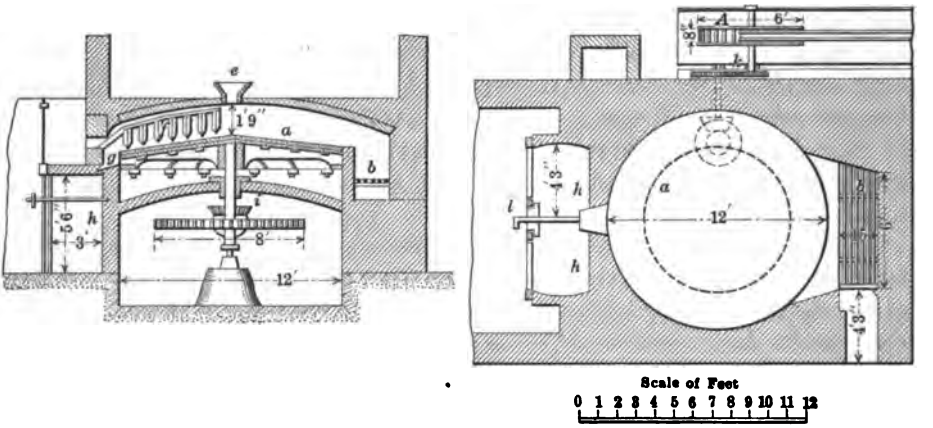
88. Rough Roasting of Metallic Sulphide for Blast Roasting.—Originally rough roasting of sulphides preparatory to blast roasting was confined to the Huntington-Heberlein process; later, with the enlargement of the field of blast roasting, the preparatory treatment was practised independently of the H. & H. process. It therefore deserves a separate discussion.

The aim is to reduce by one-half the S-content of the mixture, which usually contains from 20 to 22 per cent. S.

The furnaces used at first have been some modification of the Brunton Revolving-hearth Furnace designed for the treatment of cassiterite concentrates. In Europe the Huntington-Heberlein Furnace, a flat circular hearth supported by a shaft and large-size rollers, and driven by chain and sprocket-wheel was used in connection with the H. & H. process. In the United States the Godfrey Furnace is common; in a few instances the Holthoff Revolving Hearth has been substituted; recently the Wedge Furnace has become the rival of the Godfrey, and this especially in the rough roasting of matte. The discussion will be confined to these four forms.

89. Brunton Furnace.[1]—The furnace is represented in Figs. 131 and 132. It is a reverberatory furnace with circular rotating hearth, *a*, fireplace, *b*, central feed-hopper, *e*, fixed rabble-teeth, *g*, discharge-door, *g*, and receiving bin, *h*.



FIGS. 131 and 132.—Brunton furnace.

The slightly conical hearth, *a*, 12 ft. in diameter, is of boiler-iron, strengthened with radial ribs and covered with brickwork; it is carried by a vertical shaft which is rotated at the rate of 1 r.p.m. through a cog-wheel and bevel gearing, *i*; the shaft of the bevel pinion is connected to that of the water-wheel, *A*, through two meshing spur-wheels. The flue which carries off the gases is not shown, the chimney is indicated. The ore, fed through hopper, *e*, strikes the blades of the fixed rake, *f*, is turned over as the hearth revolves, and moved gradually from the center to the periphery to be discharged

[1] Moissenet, *Ann. Min.*, 1858, XIV, 223; *Berg. Hüttenm. Z.*, 1859, XVIII, 355.
Balling, C. A. M., "Die Metallhüttenkunde," Springer, Berlin, 1885, p. 517.

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through the door, *g*, into one of the two receiving bins, *h*, the direction being governed by the position of the deflecting slide, *l*. The furnace treats 7,200 lb. cassiterite concentrate in 24 hr. with a fuel consumption of 14.5 per cent. coal, the temperature being kept low.

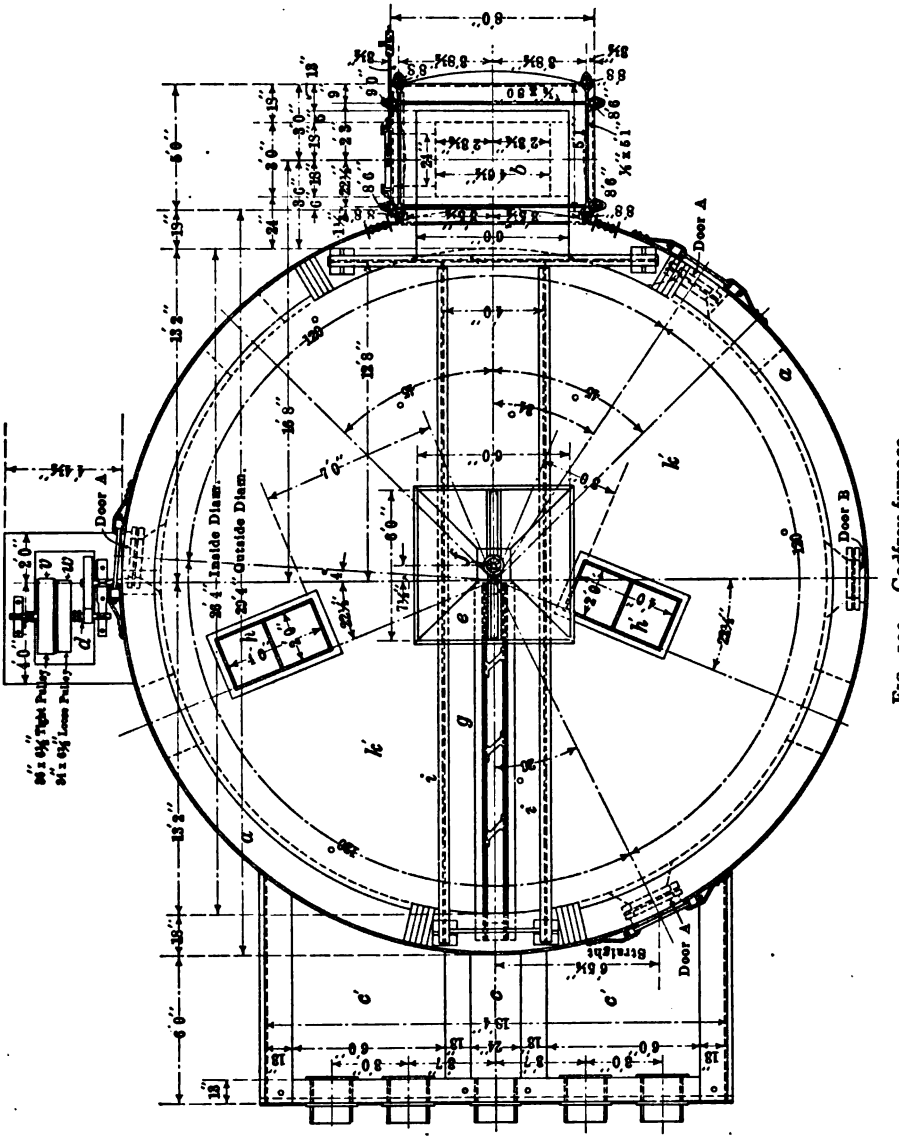

FIG. 133.—Godfrey furnace.

90. Godfrey Furnace.—This furnace is represented in Figs. 133–137 as constructed by the Stearns-Rogers Mfg. Co. of Denver, Colo., for the leading smelteries of this continent using the Huntington-Heberlein process. The drawings bring out clearly the great changes that have been made in the original de-

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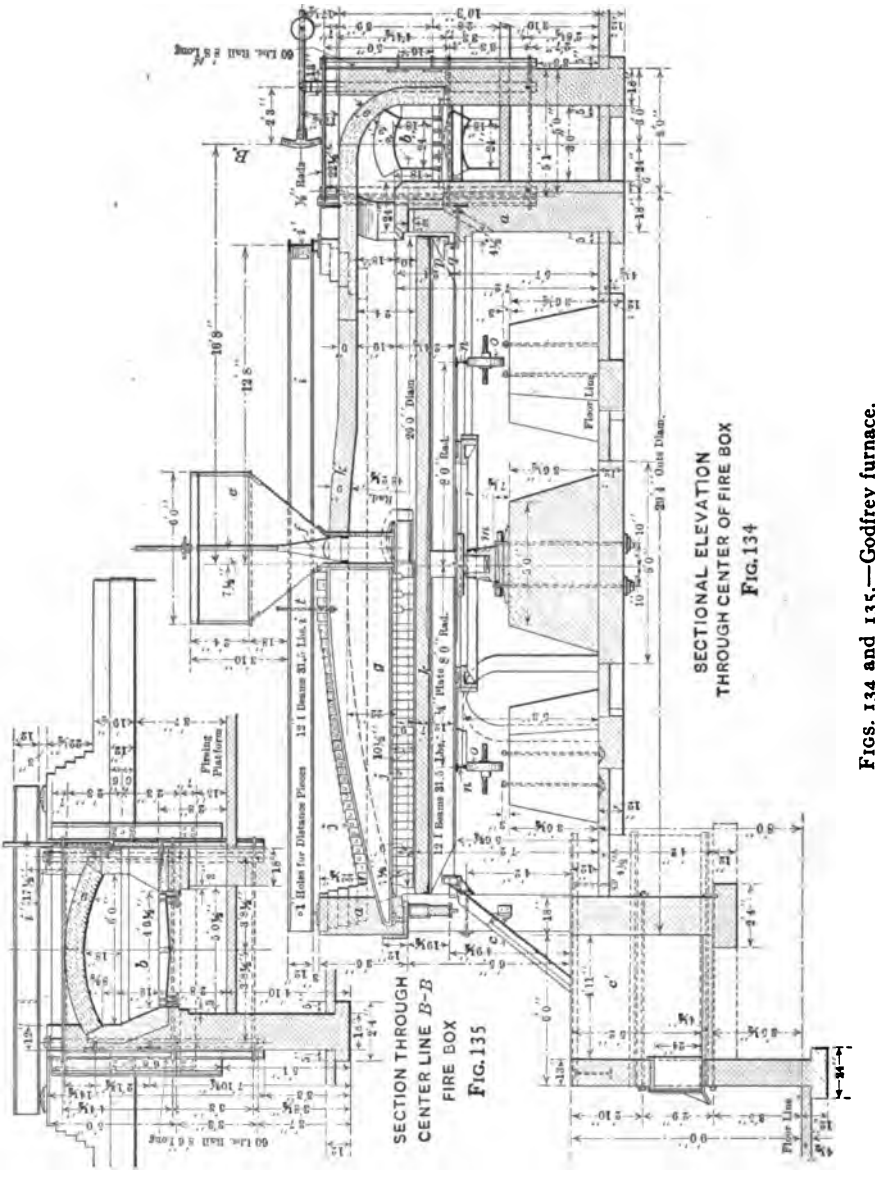
sign of Brunton. The plan, Fig. 133, shows the annular wall, *a*, of the furnace, 29 ft. 4 in. and 26 ft. 4 in. in diameter, and three doors, *A*, to furnish access to the hearth; the fireplace, *b*, 4 ft. 6½ in. by 3 ft., on the right side; chute, *c*, and stor-



FIGS. 134 and 135.—Godfrey furnace.

age bin, *c'*, for roasted ore on the left; driving shaft, *d*, with tight and loose pulleys, *v* and *w*, at the upper, and door, *B*, at the lower side entering beneath the hearth. On the roof, *k'*, are seen the ore-hopper, *e*, with feed-cone, *f*, cast-iron girder, *g*,

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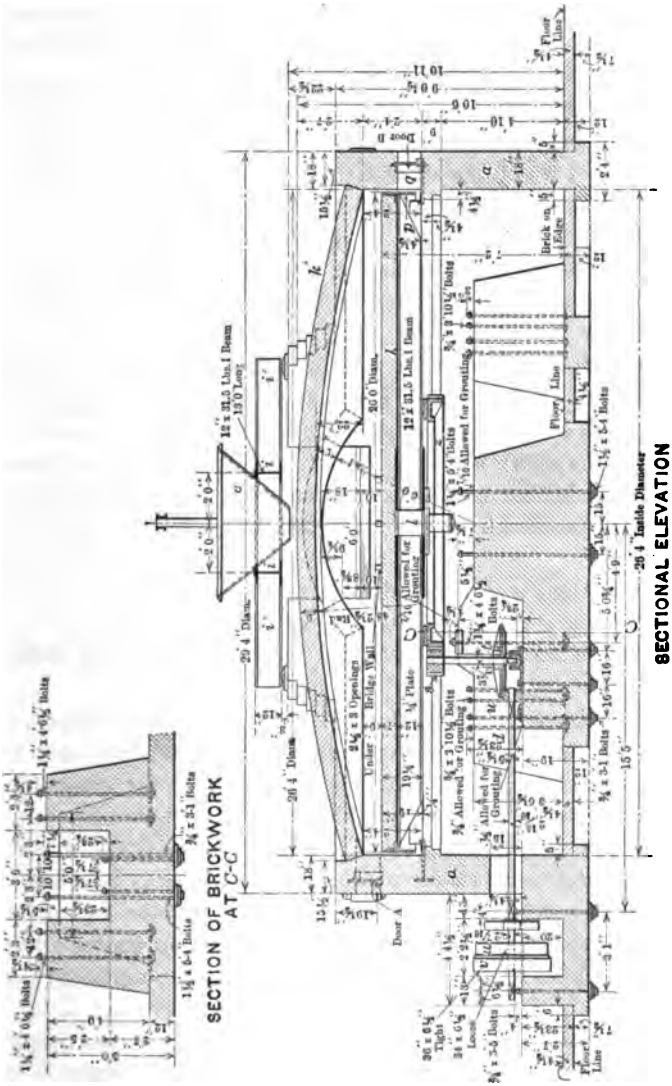
which carries cast-iron rabble-teeth, *j* (shown in Fig. 134); two flues, *h* and *h'*, for the furnace-gases, and two pairs of 12-in. I-beams, *i* and *i'*, serving as supports for girder, *g*, and ore-hopper, *e* (Fig. 136). The leading features of Fig. 134, the sectional elevation through the center of the fire-box, not seen in Fig.



FIGS. 136 and 137.—Godfrey furnace.

133, are the suspension of girder, *g*, from I-beam, *i*; the 34 stationary cast-iron rabbles, *j*, 39 by 5 in.; the brick hearth, *k*; and the roof, *k'*. The rabble-blades are spaced by a casting at the base of the girder (not shown) having suitable oblong openings, 2-in. centers; the upper parts of the blades are held in place by being clamped, some 6 to 10 in. from the ends, to round iron bars, *x*, spaced also 2-in.

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centers, with washers and nuts. The circular opening near the top of a rabble furnishes a means of inserting an iron hook for raising and lowering.

The hearth, built of 7-in. brick on a ¼-in. steel plate strengthened by 12-in.-radial I-beams, is supported at the center by spindle, *l*, revolving in step-box, *m*. At a distance of 8 ft. from the center it is carried by an inverted circular rail, *n*, traveling on four rollers, *o*; at the circumference is flange, *p*, which reaches into sand-seal, *q* (see also Fig. 136). To the I-beams strengthening the steel bottom-plate of the hearth is attached cog-wheel, *r*, which is driven by pinion wheel, *s*, as shown in Fig. 136 (sectional elevation parallel with the driving-shaft, *d*). The pinion-shaft carries at the lower end bevel-gear, *t*, meshing with bevel-pinion, *u*, the driving shaft of which, *d*, carries tight and loose pulleys, *v* and *w*. Fig. 136 shows also the manner of support of feed-hopper, *e*, which is angled to I-beam, *i*. It will be noticed that hearth, *k*, is placed high, 7 ft. 2 in., above the floor-line, in order to permit discharging into bin, *c*, Fig. 134, and delivering from this into cars which are to be hauled to the ore-beds or blast roasting apparatus.

The weight of the iron work of a furnace lies between 45,000 and 50,000 lb. and costs f.o.b. Pueblo, Colo., from $2200 to $2500. For the erection there are required 45 cu. yd. concrete, 4700 red brick, and 2000 fire-brick. The cost of erecting eight furnaces was: Excavation and filling, $900; brick-work and concrete, $9600; iron work and erection, $28,000; or $38,500 in all. This makes the approximate cost of a furnace set up $4800 (before the present advance in prices).

The general arrangement of a Godfrey plant in connection with the Huntington-Heberlein process is given in Figs. 145–147.

Another form of Brunton furnace has been described and illustrated by Blakemore.[1]

In running a furnace, the ore, charged at the center into hopper, *e*, and fed through telescopic feed-tube, *f*, controlled by a hand-wheel, falls upon the hearth and forms a cone. The rabble- and feed-blades near the center remove the ore and spread it; the next following blades turn it over and move it outward so that it travels from center to periphery in a spiral curve where it is discharged by means of an outer fixed rabble-blade through chute, *c*, into bin, *c'*.

The Brunton type of furnace furnishes an excellent means, in a reverberatory furnace of comparative small hearth area, of subjecting an ore for a considerable time to a uniform oxidizing roast under conditions of temperature which are under control.

Table 45 gives data of the work the furnace does in rough-roasting sulphide materials before they are added to blast roasting charges.

The two examples given represent good average work carried on at present. It is similar to that of earlier records of Murray, Utah;[2] East Helena, Mont.;[3]

[1] *Tr. Inst. Min. Met.*, 1898–99, VII, 323.

[2] Ingalls, *Eng. Min. J.*, 1907, LXXXIX, 527, 575. Brinsmade, *Mines and Minerals*, 1907, XXVIII, 216.

[3] Newhouse, *Tr. A. I. M. E.*, 1914, XLIX, 525.

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TABLE 45.—GODFREY FURNACE

	American Smelting and Refining Co.	
Hearth, diameter, ft., in.	26–0	26–0
Hearth, revol. in min.	9½	7
Blades, no. and size	(18; ¾ × 6 in. & 2½–4 ft.)	(20; 11 (3) and 6 (4) in. wide)
Kind of iron	Wrought iron	Steel
Life	.....	1 month
ORE MIXTURE:		
Character	(5)	Ore, matte
Size, in.	0.35	5/16
Chemical analysis	(6)	(7)
Thickness of bed, in.	3–4	3–4
Tons in 24 hr.	30	30
ROASTED ORE:		
Loss in weight, per cent.	2	8.5
Sulphide-S, sulphate-S, per cent.	Total 8–10	9.4–0.6
FLUE-DUST:		
Amount, per cent.	0.2[1]	.....
Chemical analysis	(2)	(8)
GASES:		
Cubic feet per min.	7,000	4,030 (376° C.)
Per cent. SO_2, SO_3	.....	0.82—?
FUEL:		
Character	Washed bituminous coal, ¾ in.	Bituminous coal
Amount per ton raw ore, per cent.	3	9
LABOR PER SHIFT:	¼ fireman ⅛ chargewheeler ¼ trammer ⅙ oiler	⅕ fireman ⅖ feeder ⅖ trammer
Power, h.p.	1	3

(1) Includes dust from H. & H. pots. (2) See H. & H. pots. (3) Feed and discharge blades. (4) Rabbling blades. (5) Raw matte, silicious sulphide, galena, slime. (6) SiO_2 6, Fe 30, CaO 0.5, S 22, Zn 8, Pb 21, Cu 5. (7) Matte, Silica: Pb 14, Cu 11.95, SiO_2 11.32, Zn 6.7, S 19.8, Fe 37.7, CaO 1.1; Ore, Pb 16.9, Cu 1.0, SiO_2 25.0, Zn 12.2, S 24.4, Fe 14.9, CaO 2.7. (8) Pb 15.0, Cu 4.8, SiO_2 16.6, Zn 3.5, S 9.2, Fe 16.2.

El Paso, Tex.;[1] Trail, B. C.;[2] and other works.[3] McNab[4] states that at Trail the charges used to contain Pb 40–44, Fe 10–13, SiO_2 8–11, CaO 7–10, Zn < 10, S 14–17 per cent.; also that charges with Pb > 45 per cent. did not work well. The furnaces ran slowly, at the rate of 1 rev. in 3 min., halved the S-content, and put through in 24 hr. from 38 to 45 tons charge.

[1] Vail, *Eng. Min. J.*, 1914, XCVIII, 465.
Easter, *Tr. A. I. M. E.*, 1915, LII, 716.

[2] McNab, *J. Can. Min. Inst.*, 1909, XII, 424; *Can. Min. J.*, 1909, XXX, 438, 498; *Min. World*, 1909, XXXI, 511.

[3] Hofman, *Tr. A. I. M. E.*, 1910, XLI, 739.

[4] *Loc. cit.*

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91. Holthoff Furnace.[1]—This is another form of Brunton furnace constructed by H. C. Holthoff and built by the Cudahy Power and Mining Machinery Co. of Cudahy, Wis. It was introduced at the works of the Saint Joseph Lead Co., Herculaneum, Mo., in 1907, and has been in operation since that time for the rough roasting of matte. In 1913, heating from the central gas-producer was replaced by oil-firing with modified Hauck burners which are introduced through the gas-ports, the roof of the producer having been removed. The oil in use is Kansas asphaltum-base oil of 25–26 Bé. The furnace is shown in Fig. 138. The annular roasting hearth, *a*, 36 ft. in diameter, travels at the rate of 1 rev. in 11 min. on 20 case-hardened rollers, *b*, near the circumference and 14, marked *c*, near the middle, around the central Taylor gas-pro-

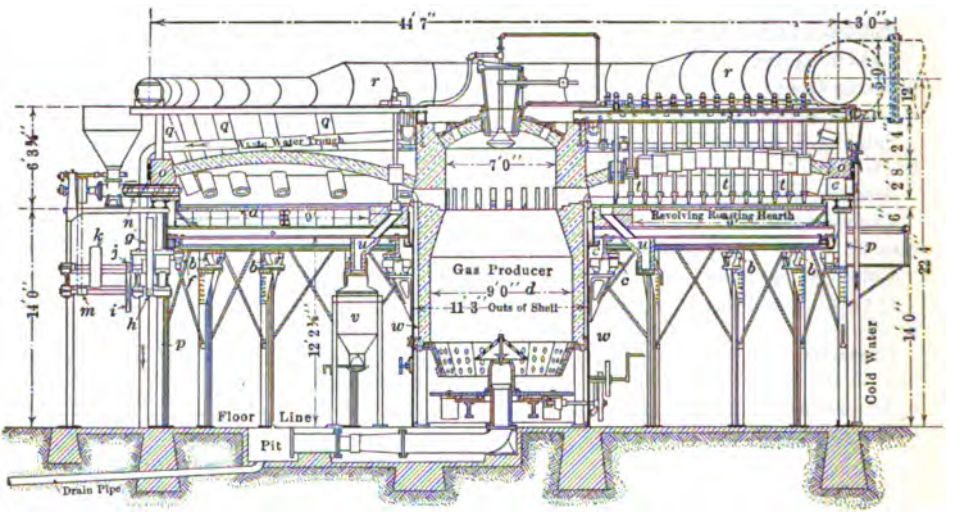

FIG. 138.—Holthoff furnace.

ducer, *d*, 11 ft. 3 in. outer and 9 ft. inner diameter, provided with a No. 3 Sturtevant blower. The hearth, 9 ft. wide, has a sand-seal, *e*, to prevent the entrance of cold air. It carries on the lower side near the circumference a circular rack driven by pinion, *f*; the shaft of the latter carries cog-wheel, *g*, meshing with pinion-wheel, *h*, whose shaft carries gear-wheel, *i*, meshing with pinion, *j*, on the main driving shaft; this receives its power through pulley, *k*, connected with a 36-h.p. motor. The shaft of wheels, *h* and *i*, carries also pulley, *m*, which is belted to the shaft of screw-feed, *n*. The roof is supported on the inner side by the brickwork of producer, *d*, which also forms one side-wall of the roasting chamber, and on the outer side by wall, *o*, carried by cast-iron columns, *p*, and bound by two 1- by 5-in. steel bands and a 1¼-in. steel cable to resist the thrust of the arch. Through the roof pass a number of tile-pipes, *q*, ending in main flue, *r*,

[1] Anon, *Eng. Min. J.*, 1905, LXXIX, 538; *Min. Reporter*, 1905, LI, 36; *Electrochem. Metall. Ind.*, 1905, III, 86; *Metallurgie*, 1905, II, 297.
Pulsifer, *Min. Eng. World*, 1913, XXXIX, 1054.
Editor, *Eng. Min. J.*, 1914, XCVIII, 900.

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which is connected with a fan and carries off the gases. Besides there pass through it four sets of stationary water-cooled rabble-arms, *s*, each with 12 blades, *t*. The ore is fed at the periphery by means of screw-feed, *n*, and moved gradually toward the center to four pockets, *u*, which travel with the hearth and discharge into two stationary hoppers, *v*, to be emptied at intervals into cars. The producer gas enters the hearth through four ports (not a series as shown in figure); it meets in the producer the necessary air, warmed by ascending through vertical flues, *w*. The flame travels radially over the hearth in a direction opposite to that of the ore.

The furnace treats in 24 hr. about 50 tons matte crushed to ¼ in., reduces the S-content from 24 to 10 per cent., makes little flue-dust, and requires about 10 tons coal or 1,000 gal. oil. The cost of the furnace is high and its repair-account is large, but its work is satisfactory.

With increase of size of plant, additional roasting capacity has been obtained by using the Wedge furnace.

92. Wedge Furnace.[1]—The furnace, shown in Figs. 139 and 140, belongs to the McDougall type of fine-ore mechanical roasting kiln, *i.e.*, it has cylindrical shaft, several superposed horizontal hearths, and a central shaft with two radial arms to a hearth provided with rabble-blades. The ore is fed mechanically at the periphery of the top-hearth which serves as a dryer, is turned over by rabble-blades and moved to the center, where it is received by feed-plates and dropped through a slot on to the first roasting hearth. Here it is moved toward the periphery that it may fall through a number of drop-holes on to the second hearth, and continue in this ziz-zag course downward, until the roasted ore is discharged from the bottom-hearth into a stationary hopper. By having a number of drop-holes at the periphery as well as near the by-pass around the central shaft, the ore passes through one drop-hole at a time and in such quantity as to fill it. While it passes through such a hole, the gases travel upward through the other holes and the by-pass around the shaft. By this arrangement the amount of dust carried along by the gases is greatly diminished.

The original seven-hearth Wedge furnace,[2] which serves for roasting pyrite and sulphide copper ores, has been modified to permit better control of temperature and thus make it suited for rough roasting of lead-bearing ore and matte, which would become sticky at the high temperature of 880° C. attained in the normal roasting of pyrite. The furnace is also provided with a fireplace on the ground-floor so as to furnish the extraneous heat necessary in roasting ores of low calorific power.

The furnace is built in two sizes. The cylindrical shell, *a*, 22 ft. 6 in. or 21 ft. 6 in. in diameter and 22 ft. 3 in. or 22 ft. 11 in. in height, is made of ½-in. steel plate and stands on structural-steel columns, *b*, 5 ft. 8½ in. high; it is lined with a full course of red brick, *c*, and has seven arched roasting-hearths with level tops, and a single drying hearth. Each roasting-hearth has two hollow

[1] U. S. Patent No. 1198882, Sept. 19, 1916. The Wedge Mechanical Furnace Co., Greenwich Point, Philadelphia, Pa.

[2] See Hofman, "Metallurgy of Copper," 1918, 112.

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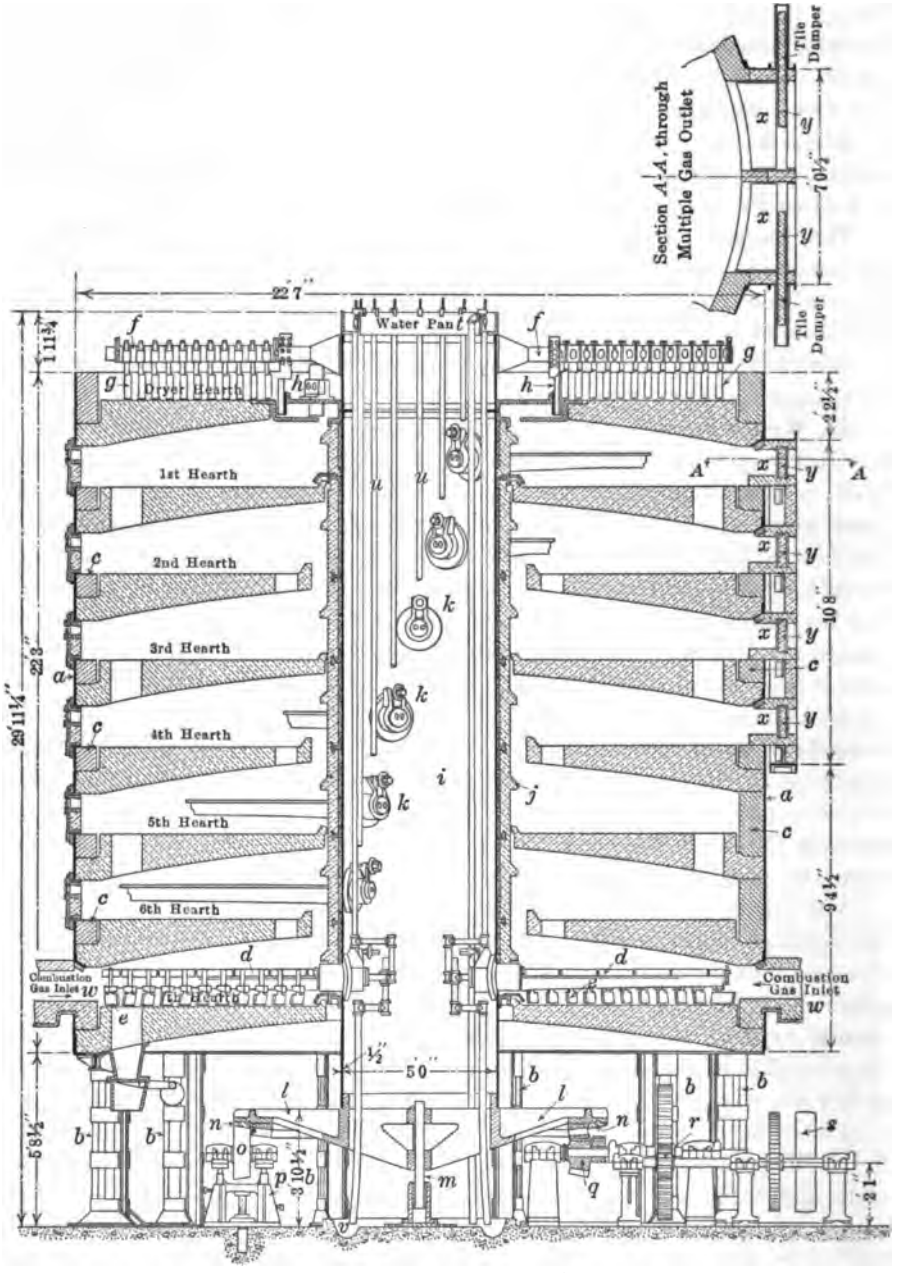

FIGS. 139 and 140.—Wedge furnace for lead ore and lead matte.

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cast-iron rabble-arms which carry cast-iron plow-shaped teeth, *e*, that are dragged over the hearth and by their weight penetrate the ore. The drying-hearth has two rabble-arms, *f*, provided with plows, *g*, which are rigidly connected and do not drag over the hearth, but are lowered when worn. They transfer the ore fed at the periphery toward the center. Here the arms, *f*, carry two or more feed-plates, *h*, which regulate the amount of ore that is to enter the furnace over a circular stationary plate; the ore forms a seal which prevents the escape of gases. The traveling and stationary feed-plates can be raised and lowered to regulate the feed and to correct the wear of the iron.

The air-cooled central shaft, *i*, of steel plate is protected from heat and gases by tongued and grooved fire-brick tiles, *j*. It is 5 ft. in diameter, which is large enough to allow a man to enter and make severally connections through cast-steel breech-locks, *k*, with the air- or water-cooled rabble-arms, *d*, of the roasting-hearths. The arms have central partitions that the air or water may flow out on one side and return on the other. The shaft is carried by master-gear, *l*, centered by pin, *m*, and provided with a race-way running on rollers which are supported by pedestals. It is rotated through the master-gear by bevel pinion, *q*, and gearing, *r*, connected with pulley, *s*, of the driving-shaft. The top of the shaft carries steel pan, *t*, which furnishes the cooling-water to the rabble-arms through separate pipes, *u*; the heated water from the rabble-arms is collected in a cement pan, *v*, in the foundation.

The two features which distinguish this furnace from the pyrite burner are two auxiliary fireplaces on the ground-floor (not shown) with combustion-gas inlets, *w*, leading to hearths 5, 6, and 7 (one only shown), and dampered gas-outlets on hearths 1, 2, 3, and 4. The main object of having the fireplaces has been noted above. It will be seen below that these fireplaces are used as a rule only in starting a furnace and in making changes from one class of ore to another, when the ore-feed is stopped for a while in order to allow the hearths to clear themselves and permit making the desired cut-off. In doing this, the temperature of the furnace drops and has to be restored by carbonaceous fuel. Some ores which do not generate enough heat to roast of their own accord when kindled may require extraneous heat for satisfactory oxidation.

The introduction of dampered outlets, *x*, had made it possible to control closely the temperatures on the upper hearths on which most of the heat is developed in roasting, and thus to permit roasting sulphides of low melting-points. In the ordinary McDougall or Wedge pyrite burner the gases ascend in the furnace in a zig-zag path of a direction opposite that of the ore. If now, as is the case here on hearths 1–4, there is added a vertical gas flue connected by four dampered horizontal passages, *x*, with the four hearths, any excess heat generated on these hearths can be bled off by pulling out the tiles, *y*, and thus the temperature regulated at will. A slight change in the position of the tiles, *y*, will show a corresponding rise or fall in the temperature measurements of the hearths.

The furnace was introduced in 1911 at the instance of A. L. Dwight at the works of the Saint Joseph Lead Co., Herculaneum, Mo., for the rough roasting

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of matte. The results were so favorable that soon other smelteries followed the example.

The metal parts of a furnace 22 ft. 6 in. in diameter weigh about 140,000 lb. and cost with the shaft-insulation $9500. The level brick arches require about 60,000 bricks (9 in.); if the hearths are leveled with concrete or other inert material, the number of brick is reduced to about 45,000; the side-walls require about 14,000 brick. The cost of erection is about $1500.

The furnace may be started in a manner similar to that of the McDougall with the exception that on account of the dragging rabble-plows crushed limestone is not fed at first to form a working bottom. A wood fire is started on hearths 3 and 5 and kept going, with a new furnace, for from 3 to 4 days in order to bring the brickwork to a dark red. Wood is frequently replaced by an oil-flame. Usually, however, the furnace is heated up from the fireplaces on the ground floor.

The ore is now fed. Its composition ranges from pyrite through mixed lead sulphide ore to galena concentrate and matte. The manner of operating therefore must show great diversity. The work done is best illustrated by citing a few examples; from these it will be possible to arrive at a few general statements.

Plant A.—The furnace is 22 ft. 7 in. in diameter, and the shell 21 ft. 3½ in. high; it requires 4½ h.p., the shaft making 17 r.p.h. The gas flues on hearths 1 and 2 are open for 5.2 sq. ft. The combustion flue on hearth 5 is open for 3.2 sq. ft. Three side-doors on hearth 7 are open, the rest are closed. The temperature on hearth 4 is 900° C., on hearth 7 about 500°. On hearths 1, 2, and 3 there is a gas pressure of 0.5, 0.03, and 0.01 in. water; this changes on hearths 4, 5, 6, and 7 to a depression of 0.02, 0.06, 0.10 and 0.15 in. water. The ore is an iron sulphide $<\frac{3}{8}$ in., with SiO_2 20, Fe 32, CaO 1, Zn 3, Pb 1, S 37 per cent. The thickness of bed is 5 to 7 in., the duration of roast about 6 hr. The loss in weight is about 18 per cent. with a reduction of S-content of about 18 per cent. There is made about 2 per cent. flue-dust. The furnace treats in 24 hr. 50 tons of ore reducing the S-content to 10 per cent. There are required 42 lb. good-grade bituminous slack per ton of ore, and three men in 24 hr. excluding wheelers for raw and roasted ore.

Plant B.—The furnace is 22 ft. 6 in. in diameter and 25 ft. 9½ in. high, requires 3½ h.p., the shaft making 28 r.p.h. The gas flue on hearth 1 is open for 8.1 sq. ft., that on hearth 2 for 5.2 sq. ft., those on hearths 3 and 4 are closed. The combustion flues are closed. The side-doors on hearths 1, 2, and 3 are closed, that on hearth 4 is open 6 in., those on hearths 5 and 6 are closed, that on hearth 7 is half-open. The draft on hearths 1–7 increases from top to bottom; hearth 1 shows 0.05 in. water; hearth 2—0.068 in.; hearth 3—0.076 in.; hearth 4—0.082; hearth 5—0.097; hearth 6—0.103; hearth 7—0.132. Oxidation is visible on hearth 2, and disappears on hearth 5. The ore is a mixture of crude (4-mesh) and concentrated (20-mesh) iron sulphide with Insol. 13.6, Fe 34.4, Cu 0.8, Pb 2.7, S 37.9 per cent. The thickness of bed is about 4 in.; duration of roast about 4 hr.; loss in weight 17.5 per cent.; S-content in roasted ore 20.5

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per cent. (all sulphide-S); amount of flue-dust small, with Insol. 19.8, Fe 33.3, Cu 1.3, Pb 4.1, S 9.5 per cent. There pass through the furnace per minute about 10,000 cu. ft. gas; flue gas contains 4–5 per cent. vol. SO_2 and 0.3–0.4 SO_3. Bituminous slack coal is used for starting. The furnace treats in 24 hr. 68 tons ore; it is served by ½ foreman, ½ oiler, 1 fireman and 4 laborers.

Plant C.—The furnace is 22 ft. 6 in. in diameter and 24 ft. 10 in. high, requires 3.5 h.p. the shaft making 28 r.p.h. Gas flues are open on hearths 1 and 2 for 2.35 sq. ft. Oil is used as fuel. Side-doors are opened when furnace gets too hot; temperature increases on the whole from hearth 1 to 6 and decreases on hearth 7: hearth 1—482° C.; hearth 2—579; hearth 3—676; hearth 4—704; hearth 5—648; hearth 6—704; hearth 7—593° C. There is a gas pressure of ⅛ in. water on hearths 1 and 2. The ore is a microcrystalline sulphide crushed through a ¼-in. screen with SiO_2 8, Fe 21.5, Zn 12, Pb 30, S 22 per cent.; thickness of bed ranges from 0.8 to above 3 in.; duration of roast 2½ hr. Sulphur-content in roasted ore is 9–12 per cent. depending upon tonnage; of the lead in the roasted ore, 68 per cent. is PbS, 22 is $PbSO_4$ and 10 per cent. is PbO. Flue-dust per day is 0.25 tons with SiO_2 8.5, Fe 16, CaO 1.8, Zn 9.8, Pb 25.2, S 11.0 per cent. There pass through the furnace per minute about 11,000 cu. ft. gas; flue gas contains 1.6 per cent. vol. SO_2 and 0.2 SO_3. There is used 0.5 gal. oil 30° Bé. per ton of ore. The furnace treats in 24 hr. 75 tons ore, and is served by 1 furnace-man and 1 helper per shift.

Plant D.—The furnace shell is 21 ft. 7 in. in diameter and 16 ft. 4 in. in height, the two bottom hearths having no shell; it requires 5 h.p., the shaft making 26 r.p.h. Gas flues on hearths 1–4 are open 4 sq. ft., side-doors on hearths 1–5 are closed, those on hearths 6 and 7 are open. Temperatures increase from hearths 1–3 and then fall off: hearth 1—260° C.; hearth 2—535; hearth 3—620; hearth 4—535; hearth 5—455° C. Draft on hearths 1–4 is 0.5 in. water. Oxidation becomes visible on hearth 2 and disappears on hearth 6. The ore is lead matte crushed to ¼-in. size with Insol. 1, Fe 53.5, Zn 4, Cu 4, NiCo 1.5, Pb 14, S 22 per cent.; thickness of bed 6 in.; duration of roast 16 hr. There is practically no loss in weight. Roasted matte contains 8 per cent. sulphide- and 4 per cent. sulphate-S. Flue-dust formed is 5 tons per month; it is smaller than 20-mesh, and has the same composition as roasted ore. Flue gases contain 4 per cent. vol. SO_2 and trace SO_3. Fuel oil of 26° Bé. is used for starting; during run 1 gal. is burnt per ton of ore. Furnace treats in 24 hr. 50 tons matte; is served by 2 men in 8-hr. shift.

Plant E.—The furnace is 22 ft. 6 in. in diameter and 28 ft. 1½ in. in height, the length of shell is 22 ft. 2½ in., requires 8.75 h.p., the shaft making 17 r.p.h. Gas flues are open for 7 ft. 1 in. Fuel is used only in starting. Two side-doors open on a hearth. Temperature rises from hearths 1 to 4 and 5, and then falls: hearth 1—98° C.; hearth 2—312; hearth 3—532; hearth 4—543; hearth 5—543; hearth 6—496; hearth 7—404° C. Draft is greatest on hearth 1 and decreases to hearth 7: hearth 1—0.080 in. water; hearth 2—0.070; hearth 3—0.065; hearth 4—0.055; hearth 5—0.055; hearth 6—0.050; hearth 7—0.045. The ore is a leady sulphide, from 5/16 in. to fine concentrate with SiO_2 18.1, Fe

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16.3, Zn 10.0, Cu 1.0, Pb 23.3, S 27.0 per cent., Ag. 8.1 and Au 0.027 oz. per ton; thickness of bed 2.5–3 in.; duration of roast 6 hr.; loss in weight 12.5 per cent.; roasted ore contains 12.5 per cent. sulphide- and 0.47 per cent. sulphate-S. Flue-dust formed is 0.71 per cent. with Pb 11.59 and Cu 1.81 per cent., Ag 10.67 and Au 0.058 oz. per ton. There pass through the furnace per minute 11,200 cu. ft. gas at 471° C.; flue gas contains 4.8 per cent. vol. SO_2 and 0.00985 SO_3. Furnace treats in 24 hr. 90 tons ore with 27 per cent. S, reducing this to 11 per cent.; with matte the daily roasting power is 75 tons; attendance 2 men per shift.

The ore delivered from the furnace drops into a vertical cylinder placed above a revolving horizontal table, and is moistened with water, thus avoiding the formation of dust. From the table the moistened ore is mechanically raked into a car.

Plant F.[1]—The furnace, 21 ft. 6 in. in diameter and 23 ft. 5 in. high, rough roasts in 24 hr. 75–80 tons lead-copper matte, crushed through 4-mesh screen, without any extraneous fuel; reduces the S-content from 22 to 5 per cent.; makes less than 8 per cent. flue-dust; requires 8 gal. cooling-water per min. entering at 4 and leaving at 21° C. About 12,000 cu. ft. gas pass off per min. Draft is small: depression in main flue 0.1 in. water; on hearths 1 and 2 it is 0.035 in.; on hearth 3—0.040 in.; on hearths 4 and 5 it is scarcely measurable.

Summary.—From the records of six plants it is seen that the manner of operating varies with the character of the ore. The leading factor is the proper control of temperature; the temperature must be low with leady sulphides and does not exceed 700° C. on the third or fourth hearth. If it becomes too high, the leady materials soften and adhere to the hearths, from which they have to be removed by barring, as was the case in the mechanical reverberatory furnaces which did not permit a satisfactory adjustment of heat. This control is obtained mainly by the correct admission of air, by opening the gas flues, and by the revolutions of the main shaft. The air necessary for oxidation is admitted on the bottom hearth, to some extent also on the next-following. With ore, only the gas flues on hearths 1 and 2 are opened; in some cases the gases are bled off by having the top flue wide open, the second two-thirds, and the third one-half. With matte, where there is much danger of fritting; all the gas flues are opened somewhat, and this to a greater extent than with ore. In heating the furnace, flame may be admitted through the three combustion inlets, *w*; usually however it enters the fifth or sixth hearth. The number of revolutions of the shaft is smaller with matte than with ore. The draft on the hearths is often so weak that it is difficult to make a measurement; in the main flue the depression ranges from 0.1 to 0.4 in. water. The drying hearth on top of the furnace removes only from 6 to 7 per cent. water from a concentrate on account of the low temperature at which the roast is carried on. The first roasting hearth drives off the remaining water and heats the ore so that it can kindle on the second; the temperature reaches a maximum on the third and in some cases on the fourth hearth, and then decreases. The tonnage ranges from 60 and 90 tons sulphide concentrate

[1] See also Anderson, *Eng. Min. J.*, 1914, XCVIII, 51.

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(SiO_2 5–18, Fe 16–18, Zn 7–10, Pb 20–24, S 27–30 per cent.) with 12 per cent. S in roasted ore, to 75 tons matte (Pb 15, Cu 12, Fe 51, S 22 per cent.) with 12 per cent. S in roasted matte. If concentrates contain more than 28 per cent. S it is advisable to dilute them with other material so that the S-content may not exceed this figure. The thickness of bed varies from 3 to 7 in., and is likely to be nearer 3 than 7. The amount of flue-dust formed varies from 2 to 5 per cent., 3 being a fair average. The gases issue from the furnace with a temperature varying from 225 to 300° C.; they carry about 3 per cent. vol. SO_2 and only a little SO_3. The fuel necessary is little; the labor required depends upon the number of furnaces and the local arrangements for handling materials; the power varies from 3 to 5 h.p.

A study of the work done by the different forms of McDougall furnaces[1] will show the changes in mode of operating necessary for treating materials that are readily fusible.

93. Huntington-Heberlein Process.[2]— This process in its original form (§84) comprises five operations: preparing of crushed ore, mixing of sulphide lead ore and limestone, rough roasting in a reverberatory furnace, moistening rough-roasted mixture, forcing air through it in a pot, and discharging blast roasted charge.

The process originally intended for a galena concentrate has been changed in many instances to meet new conditions. Thus, the presence of limestone having been found not to be essential, its addition has been dispensed with, and stress laid mainly upon the chemical composition and the physical condition of the charge. The mixtures at present are made up of oxidize fines, silicious sulphides, concentrates, slimes, and rough-roasted high-S ores; flue-dust[3] appears to interfere to some extent with the process.[4] Facts regarding these points are contained in Table 46.

Rough roasting of mixtures, frequently prepared by bedding the components, has been discussed in §88–92. The work of the Godfrey furnace (§90) merits special attention, because this furnace as well as its counterpart, the Huntington-Heberlein, was formerly used exclusively for rough roasting.

The usual way of adding the water necessary for moistening the rough roasted mixture is to have a spray of water, issuing from a horizontal pipe with numerous perforations on the lower side, sprinkle the ore as it drops from the furnace into the receiving bin, from which it is transferred either into cars or by means of a conveyor or elevator into the feed-hoppers of a battery of blowing-pots; it may however go first into a brick storage-bin which serves the battery. Figs. 133–137 show such an arrangement.

The cast-iron pots originally used by Huntington and Heberlein were conical, carried detachable hoods, and held a little over a ton of charge; they were

[1] Hofman, "Metallurgy of Copper," 1918, 91.

[2] Huntington-Heberlein, *Eng. Min. J.*, 1906, LXXXI, 1005; German Patent No. 95601, 1897; U. S. Patent No. 600347, March 8, 1898.

[3] Palmer, *Tr. A. I. M. E.*, 1914, XLIX, 511.

[4] Norton, Ridell, Dwight, *loc. cit.*, p. 485 and foll.

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mounted on trucks and trundled to the reverberatory furnaces to receive their charges. The size of pot was soon doubled, and then further increased to hold charges of 8 tons at Port Pirie, N. S. W,[1] and 10 tons at Tarnowitz, Silesia,[2] Chillagoa, Queensland[3] and other localities. The large-size pots were mounted on trunnions and tipped by worm and hand-wheel; the hoods were fixed in the building.

In the United States the pots are always of cast iron and have a capacity of from 8 to 12 tons charge; they are close on to 9 ft. inner diameter and 4 ft. 8 in. deep. In the older plants and in small works the pots are mounted on trunnions 9 ft. and more above the floor, and dumped with worm and hand-wheel.

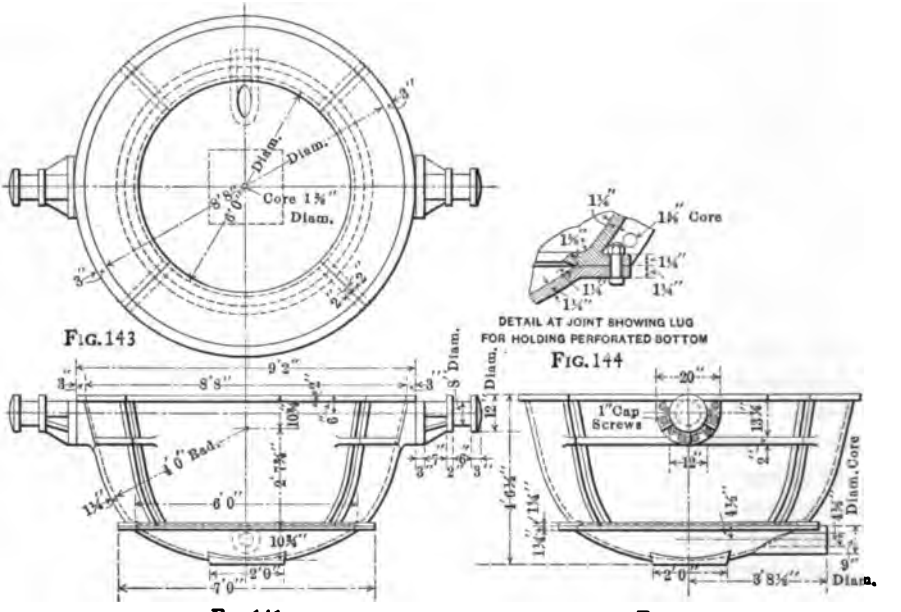

FIG. 141 FIG. 142

FIG. 143 FIG. 144

FIGS. 141 to 144.—Sectional cast-iron pot for blast roasting.

In modern large-size smelteries they rest on piers either on the furnace floor or in a pit, and are manipulated by means of an overhead crane. Originally the pots were single castings; they cracked frequently owing to the severe thermal and mechanical strains to which they are subjected. At present pots are cast in flanged sections, the bottom and four side-sections are bolted together, and the joints are made tight with asbestos packing. Such a pot is shown in Figs. 141–144, which require no comment. The blast-inlet pipe, from 4½ to 6 in. in diameter, entering tangentially at the bottom delivers pressure-air beneath a circular cast-iron plate, 6 to 8 ft. in diameter, with 200 ± holes

[1] Delprat, *Tr. Austr. Instit. Min. Eng.*, 1907, XII, 1.

[2] Birnbaum, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1905, LIII, 219; *Eng. Min. J.*, 1905, LXXX, 535, 680.

[3] Williams, *Eng. Min. J.*, 1909, LXXXVII, 603.

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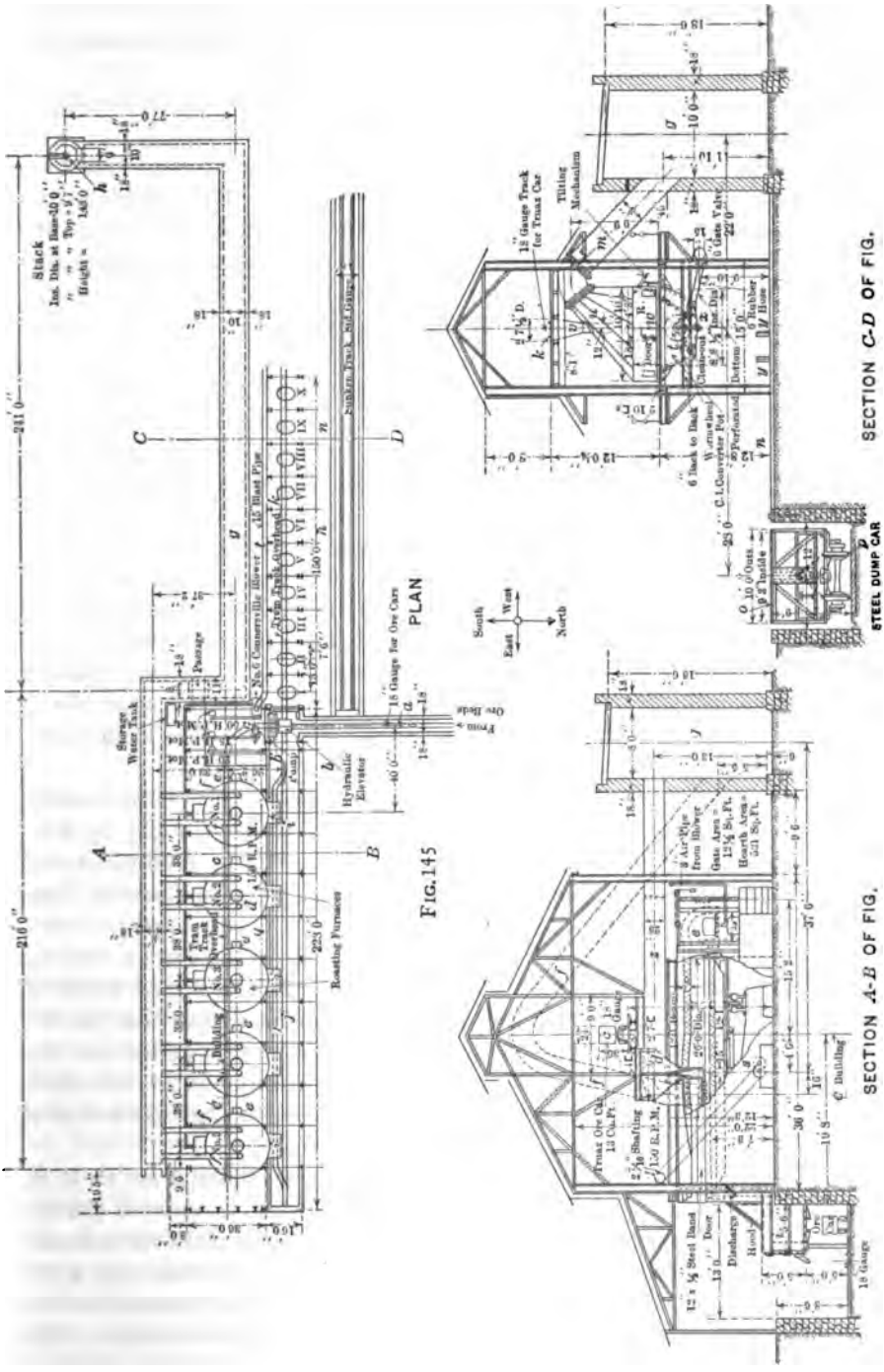

FIG. 145

FIG. 146

FIG. 147

FIGS. 145 to 147.—Huntington-Heberlein plant with Godfrey furnaces and pots tilted by worm and hand-wheel.

⅜ to ¾ in. in diameter. The plate, usually called grate, was at first cast in one piece; at present it is made up of from 4 to 6 rabbeted sectoral pieces which are supported at the center by a column about 18 in. high.

Two examples may serve to show the general arrangement and the details of a modern Huntington-Heberlein plant.

1. *Huntington-Heberlein Plant with Godfrey Furnaces and Pots Tilted by Worm and Hand-wheel.*—This is shown in Figs. 145–147. Fig. 145 gives the general plan. The roasting division has five Godfrey furnaces, Nos. 1–5, 29 ft. 4 in. outer diameter, placed at 38-ft. centers; the blast-roasting department has 10 H. & H. pots, Nos. 1–10, 8 ft. 8½ in. in diameter, placed at 15-ft. centers; between the two lies the machinery section. The total length of plant is 228+150=378 ft., and the width 100 ft.

The ore arrives from the ore-beds in Truax cars (Fig. 146) on track, *a*, Fig. 145; is raised by hydraulic plunger elevator, *b* (run, 33 ft.; speed, 60 ft. per min.; load, 500 lb.) and moved over track, *c*, to be discharged into feed-hoppers, *d*, of the Godfrey furnaces. To the right of each of these are the fireplaces, *e*. The gases pass off through gable-shaped pipe flues, *f*, into main flue and dust chamber, *g*, leading to chimney, *h*. The rough-roasted ore is discharged into bins, *i*, emptied from bottoms and sides into ore-cars running on tracks, *j* and *j'*, to hydraulic elevator, *b*, which raises the material to the level of overhead track, *k*, from which it is fed into blast roasting pots 1–10. The gases from the pots pass at the back through gable-shaped necks into main flue, *g*. The blast roasted cakes are dumped at the front on to platform, *n*, where they fall upon block, *y*, Fig. 147, and are further broken by hand, and the lumps then loaded into steel cars, *o*, on sunken track, *p*.

The machinery division has a 50-h.p. motor to drive a No. 6 Connersville blower furnishing blast to the pots; a 15-h.p. motor to drive a 5½ by 8-in. triple Gould pump to provide pressure-water to tank, *b'*, 4 ft. in diameter and 12 ft. long, and a 20-h.p. motor drive through pulley, 76 in. in diameter, line-shaft, *g*, 2 7/16 in. in diameter, making 150 r.p.m., which is connected by transmission, *r*, with the five Godfrey furnaces. Fig. 146 gives the vertical section, *AB*, of Fig. 145 through the fireplace of a Godfrey furnace, and gives details of the latter. These have been touched upon above, and are clear from the discussions given in connection with Figs. 133–137. It may be added that the gases from furnaces Nos. 2, 3, and 5 pass through horizontal flue, *z*, into main flue, *g*, while those from furnaces Nos. 1 and 4 travel through gable-shaped flue, *f*.

Fig. 147 gives the vertical section, *CD*, of Fig. 145 through an H. & H. pot. The pot, 8 ft. 8½ in. in diameter, is suspended below its center of gravity by trunnions. It is tilted in a vertical plane from the working-platform by means of worm-gear, *s*, which consists of an endless screw with hand-wheel and a spirally toothed wheel. The pot has a sectional grate, *t*, beneath which enters the 6-in. blast-inlet connected by means of a rubber hose with the blast-main. The space beneath the grate has a clean-out door, *x*. The pot is covered with sheet-iron hood, *u*, provided with feed-opening, *v*, to receive the charge, and with doors,

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w, for watching the progress of a blow and for poking down blow-holes that are likely to form.

2. *Huntington-Heberlein Plant with Godfrey Furnaces and Pots Manipulated by Electric Crane.*—Fig. 148 gives a cross-section showing the leading features. Ore and coal arrive on 16-inch belt-conveyor, *a*, which delivers into bins (ore-bins, *b*, only are shown) whence the materials are drawn into cars to be discharged into Godfrey furnace, *c*, or coal-bins, *d*, which serve the fireplaces. The power necessary for driving the machinery is derived from main shaft, *e*, and transmitted through belting to the conveyor and the Godfrey furnaces. From these the

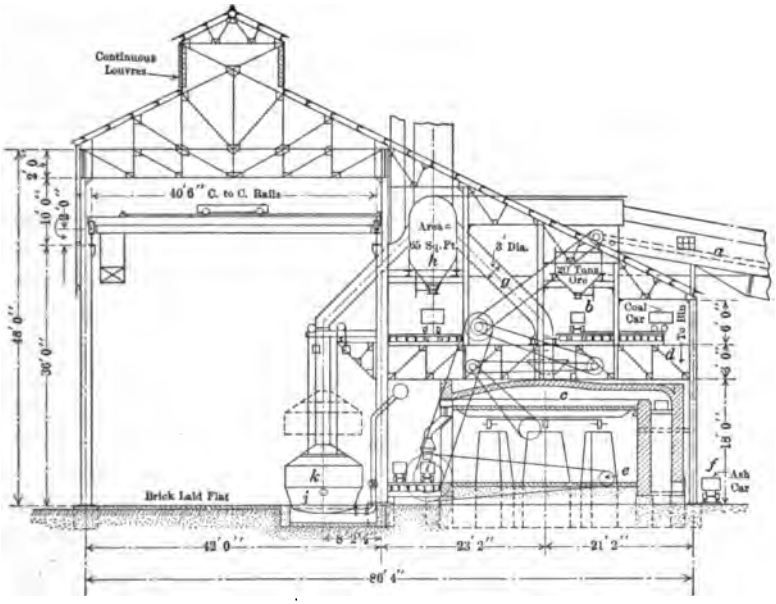

FIG. 148.—Huntington-Heberlein plant with Godfrey furnaces and pots manipulated by electric crane.

ashes are hauled away in cars, *f*; the gases pass off through pipes, *g*, into balloon flue, *h*; the rough-roasted ore is discharged into cars, *i*, which deliver their contents to pots, *j*. A series of 15-ton pots, one to three roasters, is placed in a pit served from an overhead 20-ton electric crane. The pots, 8 ft. 8 in. in diameter and 4 ft. 5¼ in. deep, are made up of 4 side-pieces and one bottom-piece; they rest on floor-blocks. The movable hoods, *k*, are balanced and deliver the gases through telescope joints to stationary pipes ending in balloon-flue, *h*. A pot is charged from a platform and transferred to its place, the hood is lowered, and blowing started. After a blow, the hood is raised, the pot, lifted by the electric crane, transferred to one end of the building, and there, by means of hook and chain, overturned from its elevated position to discharge the cake, which falls upon a platform of rails or I-beams. Frequently the fines lying on the top of the cake are poured off in a separate place while the pot travels from pit to dumping-platform. The larger pieces of blast roasted cake are picked up by

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the crane and dropped again; the smaller are lifted by hand or shoveled into a Blake crusher which reduces them to the proper size for the blast furnace.

The mode of operating a pot varies as regards detail in different plants, but is practically the same in its general features.

For the protection of the grate some works give a layer of 2 or 3 in. of ashes, crushed silicious ore, or limestone before they introduce the kindling; others begin with kindling. It used to be more common than it is at present to spread a barrow-load of ashes in a thin layer and follow this with a charge of hot calcines which served as kindling. Instead of ashes, calcareous or silicious ore free from Pb is frequently given at first. With charges low in Pb or charges which do not sinter readily the use of hot calcines forms a quick and efficient method of starting. With high-lead charges there is danger of the bottom of the charge sintering and thus obstructing the passage of the blast. Therefore hot calcines are not used in such cases, but half-burnt coal from the Godfrey furnaces. Wood serves as kindling where hot coals are not available. At El Paso, Tex., two Godfrey furnaces[1] are fired with oil, and one with coal in order to furnish the ignition fuel for the pots. The main consideration is to bring the charge to the kindling temperature without causing it to cake.

With the kindling burning in place and the blast started, from 1 to 2 tons ore-charge is given, and the blast roast started with blast pressure of from 1 to 2 oz. per sq. in. When this is under way, the rest of the charge, from 8 to 9 tons, is introduced, and the blast pressure raised, from 4 to 6 oz. at a time, until the maximum of 12 or 14 oz. is reached. Toward the end of a blow the pressure is generally lowered in order to prevent an excessive loss by dusting.

When a charge has been ignited, the oxidation with the sintering progresses upward. The sintering is probably caused by the union of SiO_2 and PbO with other components such as CaO and FeO, perhaps also of CaO and Fe_2O_3.[2] The temperature[3] attained reaches 1000° and 1200° C., and falls off quickly when oxidation and sintering cease. Thus the sintered charge beneath the rising zone of oxidation cools, and does this quickly owing to the blast passing through it.

The gases from the pots contain little SO_2; the volume of gas set free at first is small, increases gradually, and finally falls off quickly. It has always been held that the SO_2 could not be utilized in the manufacture of H_2SO_4. According to Wislecenus[4] SO_2 is converted into SO_3 by the contact process at a German plant, the pots being run in consecutive order to furnish a uniform percentage of SO_2.

When a charge has been blown, either the pot with cake and fines is dumped on to a brick pavement beneath a battery of pots, as in Fig. 147, whereupon the

[1] Vail, *Eng. Min. J.*, 1914, XCVIII, 465.
Easter, *Tr. A. I. M. E.*, 1915, LII, 716.

[2] Kohlmeyer-Hilpert, *Metallurgie*, 1910, VII, 194, 225.
Kohlmeyer, *loc. cit.*, p. 289.

[3] Hofman-Reynolds-Wells, *Tr. A. I. M. E.*, 1907, XXXVIII, 126.

[4] "Ueber die Grundlagen technischer and gesetzlicher Maassnahmen gegen Rauchschäden," Parey, Berlin, 1908, 45.

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cake is broken with sledges, and the pieces with fines are transferred to a crusher; or the pot is raised from its supports by an electric crane, as in Fig. 148, transferred to near one end of the building, where the fines are poured off, and then to the end proper, where is situated the breaking floor. This usually has a large-mouth crusher which receives lumps of suitable size.

TABLE 46.—HUNTINGTON-HEBERLEIN PROCESS

	Trail, B. C.	A. S. & R. Co.	A. S. & R. Co.	A. S. & R. Co.
POT:				
Sectional or single casting	Sectional	Sectional	Sectional	Sectional
Diameter, ft., in.	8–11 inner	10–9 outer	8–8 inner	8–8 inner
Depth, ft., in.	4–8	2–10 & 2–0	4–5¼	4–6
Blast inlet, diameter, in.	6	5	4½	5
Suspension, manner of	Rests on concrete piers	Trunnions	Rests on floor blocks	Concrete pit
Power for handling	Electric crane	Electric crane	Electric crane	Electric crane
General arrangement of series	4 rows, 9 pots in a row	3 pots in a row 16-ft. centers	?	2 rows, 12 pots in a row
GRATE:				
Sectional or single casting	4 sections	6 sections	6 sections	6 sections
Diameter, ft., in.	6–0	8–6¾	6–0	5–11
Height of center above bottom, in.	18	20(4)	18¾	18
Holes, No. and diameter, in.	1,400, ¾	.....	2,100, ⅜ to ¾	2,600, ⅜ to ¾
CHARGE MIXTURE:				
Character	(1)	Porous	(6)	(10)
Size of particles, in.	Ore through ¼; limestone ¾	¼–1	0.355	⅛ to fine
Chemical analysis	(2)	(5)	(7)	(11)
Water, per cent.	Enough to make charge ball in hand	5–8	4–6	6–10
Weight, tons	8	10–12	8(?)	7 (dump balance)
Kindling, manner of	Slabwood	Ashes on grate; then coal fire	Hot coals from Godfrey	Hot ore
Feeding, manner of	From bin, charge mixed in roaster	Whole charge at once	Dump by means of electric crane	1,500 hot Godfrey calcines
Emptying, means of	Lift and dump by crane	Lift and dump by crane	Lift and dump by electric crane	Lift and dump by electric crane
Blast pressure, start and finish, oz.	4–8	8–16	4–12	12–16
Duration of blow, hr.	6–12	8	8(?)	8(?)
CAKE, BLAST-ROASTED:				
Loss in weight, per cent.	.....	3–4	5	9
Fines, per cent.	25	5–15	5–15	10
Sulphide-S, Sulphate-S, per cent.	Total 4.5–6.5	0.2–2.0	1.3–1.2	5.0–0.7
FLUE-DUST:				
Amount	30 pots = 50 tons per year	N. d.	0.2 per cent. (8)	0.8
Chemical analysis	(3)		(9)	(12)
GAS:				
Volume, cu. ft., per min.	61,000	1,300–1,800	800–1,000	1,600
Per cent. SO_2, SO_3	0.2/0.8-none	N. d.	N. d.	1.5–0.0052
LABOR PER SHIFT:	1 man, 3 pots	1 man, 3 pots	1 man, 3 pots	1 man, 2 pots

(1) Complex lead-zinc ore, galena conc., limestone. (2) Pb 27, Fe 18, SiO_2 8.6, CaO 8, ZnO 10.2, S 11. (3) Pb 43.6, Fe 11.0, SiO_2 3.0, CuO 4.0, S 11.3, Zn 4.8. (4) 10 in. above bottom of grate. (5) Pb 49.2, SiO_2 10.5, PeO 15.4, CaO 4.9, S 8.9, MgO 2.4, Al_2O_3. 2.4, Zn 3.0. (6) Matte, galena, flotation slime, silicious sulphide, pre-roasted Godfrey material, limestone. (7) SiO_2 6, Fe 30, CaO 0.5, S 9, Zn 8, Pb 21, Cu 5. (8) Includes dust from H. & H. (9) SiO_2 5.8, Fe 11.7, Zn 3.2, CaO trace, S 8.1, Pb 22, Cu 3, Ag 18 oz., Au trace. (10) Pre-roasted ore, matte, sulphides, silicious lead ore. (11) Pb 21, Cu 1, SiO_2 22, Zn 8.4, S 14.5, Fe 15, CaO 2. (12) Pb 21, Cu 1.8, SiO_2 3.7, Zn 4, S 15.6, Fe 15.3.

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At the works of the consolidated Mining and Smelting Co. of Canada, Trail, B. C., which have 38 H. & H. pots, a crane delivers the blown pot to the breaking floor, pours off the fines, and dumps the cake, which is broken by a drop-weight into lumps. An electric crane with clam-shell bucket picks up lump and fines, and delivers both to a horizontal bed, 11 by 8 ft., of steel rollers, 5–6 in. in diameter and 3-in. centers; these screen out the fines, which go to the Dwight-Lloyd machines, and feed the over-size to a Blake crusher, 24 by 36 in. The crushed material drops into cars which are hauled to storage-bins of cement concrete, and emptied.

Details of Huntington-Heberlein pots are given in Table 46.

Double-treatment is taken up in §100.

The cost of blast roasting of 1,000 tons or more ore in a 50-ton plant has been estimated by Austin.[1] The loss in weight was 25 per cent.; the gain in Au, 7 per cent.; the loss in Ag, 1.4 per cent.; in Pb, 8.5 per cent.; and in Cu, 1.4 per cent. An average analysis of blast roasted ore gave: SiO_2 6, Fe 35, Cu 1, Zn 8.5, Pb 15, S 5.5 per cent., and Ag. 21 oz. per ton. The cost of blast roasting was: 13 men at $2.10 per day, $27.34; foreman (part), $1.40; fuel for roasting (slack coal), 6 tons at $1.55, $9.30; power (estimated), $3.00; lighting (estimated) $0.40; repairs and supplies (estimated), $2.50; total for 43 tons, $43.94, or $1.02 per ton.

The cost of roasting in a hand reverberatory furnace at the same plant was $2.00 per ton; and of briquetting roasted material, $0.60 per ton. The loss in Pb ranged from 11 to 17 per cent.

Comparing the two processes gives: cost, $1.02 *vs.* $2.60; loss in Pb, 8.5 *vs.* 11–17 per cent.; quality of product for blast furnace, good *vs.* passable.

Another estimate of cost is that of Ingalls[2] which gives a higher figure.

94. Other Forms of Up-draft Blast Roasting Pots and Processes.—The success of the Huntington-Heberlein process prompted the construction of different forms of pots which in part called for modifications in the mode of operating.

In Fig. 148 the gases from the pot ascend through a vertical telescopic and then through an inclined pipe into the balloon flue. The telescope pipe is in the way of the crane; it is quickly corroded as is the balloon flue, the life of the pipe averaging perhaps only 6 months; the dripping moisture also corrodes the hood of the pot. These disadvantages are obviated by the Richard hood, shown in Figs. 149–151. Hood *a*, which has a larger diameter than blow-pot, *b*, rests in groove, *c*, carried by brick and concrete wall, *d*; the joint is made gas-tight by a sand-seal. From the side of the hood passes out horizontally pipe, *e*, with elbow which reaches into the sand-seal placed on top of brick-chamber, *f*; from this, uptake, *g*, carries off the gases to the balloon flue serving a series of pots. When a charge is to be introduced or the blast roasted cake to be discharged, hood, *a*, with pipe, *e*, is raised by means of an electric traveling crane and transferred to one side, and then the pot removed to the dumping-platform.

[1] *Min. Sc. Press*, 1908, XCVI, 641.

[2] *Tr. A. I. M. E.*, 1907, XXXVII, 528.

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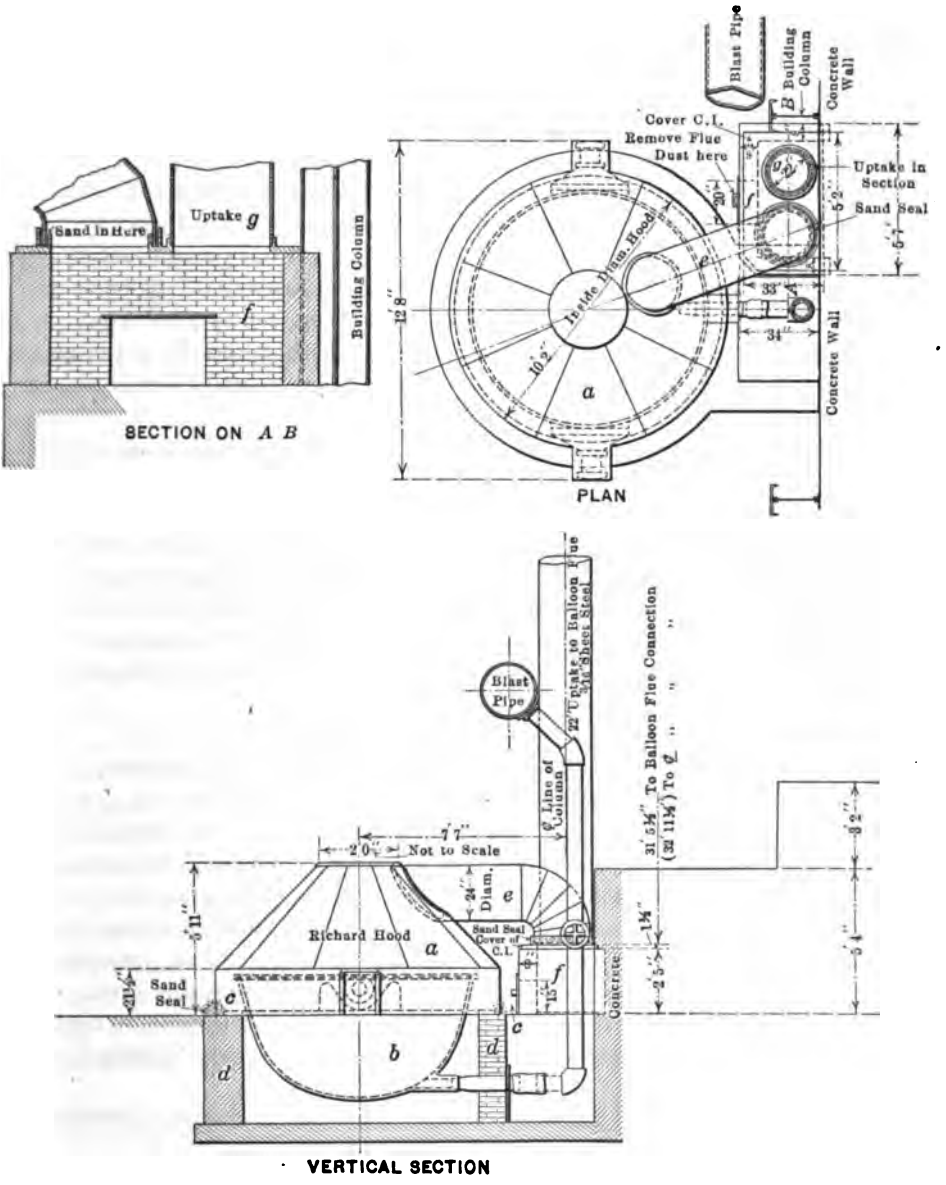
In some plants the pot with its blown charge is first placed by the crane on to platform scales to ascertain the loss in weight caused by blast roasting. The



FIGS. 149 to 151.—Richard hood.

Richard hood has replaced the usual form in several plants, and is going to do it in others.

F. D. Baker[1] patented a hood which covers the blowing pot in manner

[1] U. S. Patent No. 942810, Dec. 7, 1809.

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similar to that of Richard, but the gases set free in blast roasting are deflected by the hood, pass off between hood and rim of pot, descend outside of the latter, and enter a flue connected with an exhaust apparatus. Doors in the hood give access to the charge when the blast has been shut off. This pot, found in several Colorado plants, has the disadvantage that it has to be filled before the blast can be turned on, and does not permit correcting surface conditions of the charge without shutting off the blast, and even then much smoke passes off into the open. These pots have therefore fallen into disfavor.

H. Haas[1] constructed a pot which was in operation at Torreon, Mexico. The essential parts are a bowl-shaped, cast-iron vessel, 6 by 12 ft., with grate, holding 10 tons of charge, supported by trunnions through which enters the blast; a stationary hood (with doors) delivering the gases to a balloon-shaped dust-flue; a hopper for holding the prepared mixture supplied by a belt-conveyor; a tilting device; and an adjustable air-supply. With a 400-ton plant of 20 units, each treating two charges per day, the cost of blast roasting is estimated at 68 cts. per ton.

H. C. Bellinger[2] patented a continuous up-draft furnace which consists of a horizontal traveling grate, over part of which extends a vaulted roasting chamber connected by a pipe with a device for furnishing induced draft. At the fuel end of the grate and outside of the roasting chamber are two hoppers; one feeds the ore mixture on to the grate to be ignited by a burner; the other feeds on top of the ore a bed of crushed limestone, coarse ore, or other suitable material which, serving as a filter, holds back the dust created by the power of the induced draft. The sintered ore is discharged outside of the roasting chamber at the end opposite the feed.

Another apparatus to be mentioned is that of Vivian.[3]

Robinson Roaster-box.[4]—This has been in operation at Midvale, Utah. The leading parts are a perforated cast-iron plate forming the roof of an air-box and the floor of a low brick shaft with door at the back for admittance of rack- and pinion-driven pusher, and a sliding door at the front for passage of blast roasted cake. A row of 20 furnaces did the work of the plant at a cost of $1.20 per ton. They have been replaced by Dwight-Lloyd machines (§97).

F. G. Kelley[5] used at the Tintic smeltery, Utah, a tilting tray-shaped vessel with grate, provided with under-grate blast, and movable hood.

Knapp-Kunze[6] blast roasted raw copper-bearing galena in pots at Zeehan, Tasmania, where the operation goes by the name of the Knapp-Kunze process.

McMurtry-Rogers[7] did similar work with copper ores and matte at Wallaroo,

[1] *Eng. Min. J.*, 1910, XC, 814.
[2] U. S. Patent No. 942052, Dec. 10, 1910.
[3] U. S. Patent, No. 950798, March 1, 1910.
[4] Hofman, *Tr. A. I. M. E.*, 1910, XLI, 747.
Palmer, *Mines and Minerals*, 1909–10, XXX, 496.
[5] Hofman, *Tr. A. I. M. E.*, 1910, XLI, 747.
[6] Poole, Sydney University Eng'g. Soc., Nov. 11, 1898.
[7] Cloud, *Tr. Inst. Min. Met.*, 1906–07, XVI, 311.

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South Australia, as did Wallace with the Detroit Copper Co., Morenci, Ariz., and Browne with the Canadian Copper Co., Copper Cliff, Ont.[1]

There may be mentioned the Bormettes process[2] which consists in incorporating sulphide lead ore into fluid slag by stirring, and then blowing the mass in a pot; and the Lotti process[3] which is similar.

F. Heberlein patented[4] the agglomeration in a converter of iron oxides, manganese oxides, flue-dust, etc. by adding carbonaceous fuel to the charge, and blowing this in the usual way; photomicrographs of the agglomerated material have been prepared by Klugh.[5]

95. Carmichael-Bradford Process.[6]—The leading points of this process, which had its origin in Australia, are that sulphide lead ore is mixed with partly dehydrated gypsum; that the mixture is wetted, dried, broken into lumps; and that the lumps are blast roasted in a pot in which a small fire has been started. The products are a porous desulphurized cake, and a gas rich in SO_2 and SO_3. The process was in operation at Port Pirie, N. S. W., but has been replaced by Dwight-Lloyd machines (§97).

The gypsum, $CaSO_4+2H_2O$, is ground in a disintegrator, heated to 200° C. on an iron pan, 20 ft. long, with push-conveyor, to expel in 20 min. about 50 per cent. of the water ($CaSO_4+1H_2O$), and screened to remove lumps, which are discarded, as they run high in SiO_2. Galena and sulphide concentrates are worked with partly dehydrated gypsum in the proportion of 3 : 1 in a treater having flat revolving cones. The material dropping from the mixer is wetted and passed into a horizontal pug-mill, 6–8 ft. long and 2 ft. 6 in. in diameter, from which it issues through ¾-in. holes in the form of bars, and drops into a conical drum, 5 ft. long, 2 and 4 ft. in diameter, making 20–25 r.p.m., which breaks the bars into pieces of nut-size. These are delivered to a pan-conveyor which travels at the rate of 1 ft. per min. through a drying furnace 10 ft. long. The granulated material loses 50 per cent. of its water, comes to a set, and is ready for the converting pots.

The general average of the charge-mixture is PbS 29, ZnS 21, FeS 2, Fe_2O_3 2.5, FeO 1, MnO 5, Al_2O_3 3, CaO 10, SiO_2 14, SO_3 12; total 99.5 per cent.

The converters are conical pots suspended 6–7 ft. above the floor and tilted by a worm-gear; they have the usual grates, are 6 ft. in diameter and 5 ft. deep, have a capacity of 4 tons charge; a pot is blown through a 4-in. pipe in about 1 hr. with a blast pressure of 10 oz. The S-content of the charge is reduced from 14 to 4 per cent.; an average analysis of the gas is SO_2 0.4,

[1] Hofman, *Tr. A. I. M. E.*, 1910, XLI, 749.
[2] *Eng. Min. J.*, 1905, LXXX, 580; *Mining Mag.*, 1905, XII, 426; *Metallurgie*, 1905, II, 353.
[3] U. S. Patent No. 847017, March 12, 1907.
[4] *Oesterr. Zt. Berg. Hüttenw.*, 1908, XLVI, 555.
[5] *Tr. A. I. M. E.*, 1913, XLV, 334.
[6] U. S. Patent No. 705904, July 12, 1902.
Clark, *Eng. Min. J.*, 1903, LXXVIII, 708.
Editor, *op. cit.*, 1905, LXXX, 778, ill.
Poole, *Bull. Sydney University Eng'g Soc.*, Nov. 11, 1908.
Hofman-Mostowitsch, *Tr. A. I. M. E.*, 1908, XXXIX, 628.

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SO_2 8.5, CO_2 0.5, CO none, O 10.6, N 80 per cent. vol. In the process, fuel is fed and kindled, a gentle blast turned on, the charge introduced, and the hood lowered. The charge warms, water is driven off, then follows sulphurous gas, which is conducted into a sulphuric-acid chamber.

There has been much speculation regarding the chemical reactions in which this process differs from the Huntington-Heberlein and Savelsberg processes treated in §§93 and 96. Hofman-Mostowitsch[1] found that the decomposition temperatures of $CaSO_4$ by heat in the absence or presence of SiO_2 are higher than those attained in blast roasting; Schütz,[2] however, has shown that the decomposition of $CaSO_4$ by SiO_2 in the presence of PbS (see §85) begins at 800° C. and is complete at 1,000–1,100° C.; also that the strongly endothermic reaction requires the presence of an extraneous source of heat, which in the present case is supplied by FeS and ZnS. The supposed reaction $PbS+CaSO_4=PbSO_4+CaS$ does not take place.[3] The short time required for treating a charge may be attributed to the intimate mixture of ore and flux, and to the reaction of Schütz.

The cost per ton of ore, with labor at $1.80 per 8 hr., gypsum at $2.40 per 2240 lb., and coal at $8.40 per 2240 lb. is estimated to be: 0.25 ton gypsum, $0.60; dehydrating and granulating gypsum, $0.48; drying charge mixture, $0.12; converting, $0.24; breaking sintered cake, $0.12; 0.01 ton coal, $0.08; total, $1.64. Crediting lime in the sintered product, estimated at $0.12, reduces the cost to $1.52.

From the foregoing it appears that in principle and practice the process gives good results, provided there is a cheap supply of gypsum, and a satisfactory way for utilizing the large volume of SO_2 set free; its field is therefore limited. The reasons for giving up the process in Australia were that the sulphurous gas never gave an SO_2-content exceeding 2.6 per cent. and that the S-content of the cake averaged 6.8 per cent.[4]

96. Savelsberg Process.[5]—The leading feature of this process is that raw galena concentrate mixed with from 15 to 20 per cent. crushed limestone and silicious ore, and wetted, is charged gradually into a pot on to a glowing bed of fuel, and blown. The converter is bowl-shaped and rests upon a truck. The original pots had a capacity of from 1 to 2 tons charge, but have been increased in size to hold 8 tons.

At Ramsbeck, Westphalia, the converter is 6.56 ft. in diameter, weighs 2860 lb., and holds an 8-ton charge. A typical charge consists of 100 parts

[1] *Tr. A. I. M. E.*, 1908, XXXIX, 628.
[2] *Metallurgie*, 1911, VIII, 228.
[3] Doeltz, *Metallurgie*, 1905, II, 460; *Eng. Min. J.*, 1906, LXXXI, 175, see also §85.
[4] Riddell, G. C., private communication, Sept. 13, 1916.
[5] U. S. Patent No. 755598, March 22, 1904; German Patent No. 171215, 1906.
Ingalls, *Eng. Min. J.*, 1905, LXXX, 1067.
Savelsberg, *Min. Mag.*, 1905, XII, 391.
Editor, *Eng. Min. J.*, 1906, LXXXI, 1136.
Hofman-Reynolds-Wells, *Tr. A. I. M. E.*, 1907, XXXVIII, 126.
Hofman, *op. cit.*, 1910, XLI, 746.

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galena concentrate (Pb 60–78, and S 15 per cent.), 10 silicious silver ore, 10 spathic iron ore, and 19 limestone. The mixture contains therefore about 11 per cent. S and 9 per cent. SiO_2. Ore and flux are pulverized separately in roller- or ball-mills to pass a 3-mm. screen, are bedded, and mixed twice; before the second mixing 5 per cent. water is added. In starting a blow, the grate is covered with a layer of crushed limestone; this is followed by a bed of glowing coal, and the latter covered with limestone. The blast is started, and moistened charge fed to a depth of about 12 in. When the surface has become red hot, a second layer of mixture is introduced, and the operation continued until the converter is filled. While charging, about 250 cu. ft. of air is forced in per minute at a pressure of from 2.75 to 4.50 oz. per sq. in.; the volume is then increased until the pressure reaches 11.5–13.5 oz. The 8-ton charge is blown in about 18 hr. The converter is tilted, and the cake dumped upon a vertical iron bar; it is thus broken into several pieces which are further reduced in size by hand. Under favorable conditions the S-content of the charge is reduced to 2–3 per cent.; the loss in Pb is said to be 1 per cent. and that of Ag *nil*.

At Herculaneum, Mo.,[1] the Saint Joseph Lead Co. had in operation 18 pots placed in two rows. A pot, cast in one piece, was 8½ ft. in diameter and 4½ ft. deep, and had a 5-in. blast-inlet. The grate, cast in two pieces, was 3½ ft. in diameter and 8 in. above the bottom at center; it had thirty ½- in. holes. The pots were suspended from trunnions and tilted by worm-gear. A charge of 7 tons, consisting of galena concentrate and roasted matte ranging from ½-in. to 200-mesh, and mill-tailing, was prepared mechanically,[2] delivered by belt-conveyor to boot of elevator, discharged into bin above charging-floor and fed from there by hand to the pots. The following two analyses give the composition: SiO_2 13.12, FeO 5.90, CaO 6.60, MgO 3.30, Pb 47.40, Zn 2.10, S 11.20, H_2O 6 per cent.; and SiO_2 12, FeO 20, CaO 9, Pb 42, S 10–12, H_2O 5 per cent. In starting, glowing coal from a separate fireplace was first put in, the blast turned on, and then charge fed gradually until the pot was filled. The blast-pressure, 10 oz. at first, was raised to 20 oz., and increased to 25 oz. toward the end of the 10-hr. blow; the amount of air blown ranged from 1000 to 1500 cu. ft. per min.; from 15 to 20 per cent. of the charge remained unsintered and had to be retreated. The blast roasted cake was dumped and broken by hand; its composition was SiO_2 17.2, FeO 9.0, CaO 7.0, MgO 4.0, Pb 44.2 Zn 3.8, S 2.3 per cent. The S-content of this analysis is low, the average is nearer 3.5 per cent., of which 3.0 per cent. is sulphide- and 0.5 sulphate-S. The flue-dust formed amounted to about 2 per cent. One man in a shift looked after a pot. The charge lost about 10 per cent. in weight; the loss in Pb was about 2½ per cent. A pot furnished about 12 tons blast roasted material per day. The cost was about $2 per ton charge.

The process has been replaced by blast roasting in a Dwight-Lloyd machine, (§97) which has halved the cost. Its additional disadvantages have been

[1] Private communication by O. M. Biharz, 1910, and W. A. Smith, 1916.

[2] Editor, *Eng. Min. J.*, 1910, LXXXIX, 648.

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the requirement of adding a large amount of limestone to reduce the calorific power of the ore, and the necessity of decomposing it afterward.

The process[1] is very sensitive to slight variations in composition of charge, and requires close adherence to certain proportions of SiO_2 and PbO.

A calculation of the heat distribution in a specific case by J. W. Richards[2] shows that 22 per cent. of the heat passes off in the gases, 18 per cent. is required for the decomposition of limestone and 4 for that of the siderite, 9 per cent. for the evaporation of moisture, 23 per cent. is found in the sensible heat (@400° C.) of the product, and the difference, 25 per cent., is lost by radiation and conduction.

Laboratory experiments by Hofman-Reynolds-Wells[3] with charges weighing 20 kg. have covered the composition of charge, pressure of blast, rise and fall of temperature, and loss in Pb and Ag; see also Packard.[4]

An innovation in the original process is the admission of steam with the blast.[5] The steam is to condense in part in the upper cooler layers of the charge, to moisten them and thus prevent dusting; in part it is to be decomposed by the glowing fuel, form water gas, and thus promote sintering.

97. Dwight-Lloyd Process.[6]—This process differs from the intermittent up-draft processes so far discussed, in that it works continuously with a down-draft acting upon a thin layer of ore spread on and held in place by a traveling grate. Up-draft processes have the following disadvantages: They are intermittent; the pots require much attention; the charges may contain only a small percentage of fines; they are liable to suffer loss in Pb, as they are exposed for considerable time to the influences of hot gases; they are unevenly sintered and furnish considerable amounts of fines which cause loss by dusting and volatilization, and have to be retreated; the sinter-cakes have to be broken, which

[1] Mostowitsch, *Tr. A. I. M. E.*, 1916, LV, 741.
[2] "Metallurgical Calculations," McGraw-Hill Book Co., New York, 1918, part III, p. 586.
[3] *Tr. A. I. M. E.*, 1907, XXXVIII, 1261.
[4] *Loc. cit.*, p. 935.
[5] Savelsberg, U. S. Patent No. 1103318, July 14, 1914.
[6] Dwight, *Min. Ind.*, 1907, XVI, 380.
Hofman, *Tr. A. I. M. E.*, 1910, XLI, 755.
Dwight, *op. cit.*, 1914, XLIX, 491.
Work carried on at:
Trail, B. C.; Jacobs, *Met. Chem. Eng.*, 1911, IX, 405.
Salida, Colo.; Hough, *Min. Sc. Press*, 1912, CIV, 895.
Correspondent, *Met. Chem. Eng.*, 1912, X, 87.
Weeks, *Tr. A. I. M. E.*, 1915, LII, 711.
Tooele, Utah; Thomson-Sicka, *op. cit.*, 1913, XLVI, 946.
Herculaneum, Mo.; Pulsifer, *Min. Eng. World*, 1913, XXXIX, 1054.
Editor, *Eng. Min. J.*, 1914, XCVIII, 900.
East Helena, Mont.; Riddell, *Tr. A. I. M. E.*, 1914, XLIX, 495, 530.
Newhouse, *op. cit.*, 1914, XLIX, 525.
Murray, Utah; Norton, *op. cit.*, 1914, XLIX, 485.
El Paso, Texas; Vail, *Eng. Min. J.*, 1914, XCVIII, 465.
Easter, *Tr. A. I. M. E.*, 1915, LII, 716.

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is costly and unhealthy. The trend at present is from intermittent up-draft to continuous down-draft apparatus.

Three types of sintering machines have been constructed by the Dwight-Lloyd Sintering Co. of New York: the drum, the straight-line, and the horizontal-table machine. All embody the same three functions: a layer of well-mixed and moistened charge is spread mechanically to the thickness of 2.5 to 5, usually 4 in., on a traveling grate; the charge-stream first passes under an igniter to be kindled at the surface, and then over a stationary suction-box which by suction causes the roasting started at the surface to progress downward and be finished when the charge reaches the further end of the suction-box; the sinter formed is discharged automatically in sizes suitable for the blast furnace into a receiver.

The drum-machine[1] and the horizontal-table machine[2] have been superseded by the straight-line machine discussed in detail in §98.

Interesting developments of the table machine are those of Weeks[3] and of von Schippenbach.[4] The latter, which is 14 ft. 6 in. in diameter, withdraws the roaster gas in such a manner that rich gas, with 4–5 per cent. volume SO_2 suited for the manufacture of H_2SO_4, is continuously separated from poor gas with 0.2 per cent. volume SO_2, which goes to waste. At Cerro de Pasco, Peru, Dwight-Lloyd[5] have used a shallow tray with down-draft which worked intermittently. Other forms of down-draft apparatus are those of Plock,[6] Greenawalt[7] and Meyer.[8] Laboratory apparatus for down-draft experiments have been described by Hahn[9] and Pulsifer.[10] The application of down-draft to the sintering of oxide iron ore mixed with coal is discussed by Gayley[11] and Klugh.[12]

98. Dwight-Lloyd Straight-line Machine.—This machine is shown in Figs. 152 and 153. It consists of a frame of structural steel supporting a feed-hopper, an igniting furnace, a suction- or wind-box, and a pair of endless-track circuits to accommodate a train of small truck-like elements called pallets which, in com-

[1] Dwight, *Min. Ind.*, 1907, XVI, 380; *Eng. Min. J.*, 1908, LXXXV, 649.
Hofman, *Tr. A. I. M. E.*, 1910, XLI, 756.

[2] Hofman, *loc. cit.*

[3] U. S. Patent No. 916903, March 30, 1909; *Eng. Min. J.*, 1909, LXXXVII, 1140.

[4] Kroupa, *Oesterr. Zt. Berg. Hüttenw.*, 1912, LX, 513, 539.
Guillet, *Rev. Mét.*, 1911, VIII, 599.
Hofman, *Min. Ind.*, 1911, XX, 480.
Correspondent, *Zt. angew. Chem.*, 1913, XXVI, 599; *Min. J.*, 1913, CII, 900.
Editor, *Eng. Min. J.*, 1913, XCV, 1061.

[5] *Min. Sc. Press*, 1909, XCVIII, 195.

[6] U. S. Patent No. 1089153, March 3, 1914.

[7] U. S. Patents No. 1088818, March 3, 1914, and No. 1098035, March 26, 1914.
Austin, *Min. Sc. Press*, 1911, CII, 177.

[8] U. S. Patent No. 1166142, Dec. 28, 1915; *Eng. Min. J.*, 1916, CI, 526.

[9] *Eng. Min. J.*, 1911, XCI, 1208.

[10] *Op. cit.*, 1913, XCVI, 308.

[11] *Tr. A. I. M. E.*, 1911, XLII, 180; *Iron Age*, 1912, LXXXIX, 73.

[12] *Tr. A. I. M. E.*, 1912, XLIII, 364, 739; 1913, XLV, 330; *Iron Trade Rev.*, 1915, LVII, 835; *Iron Age*, 1916, XCVI, 1000.

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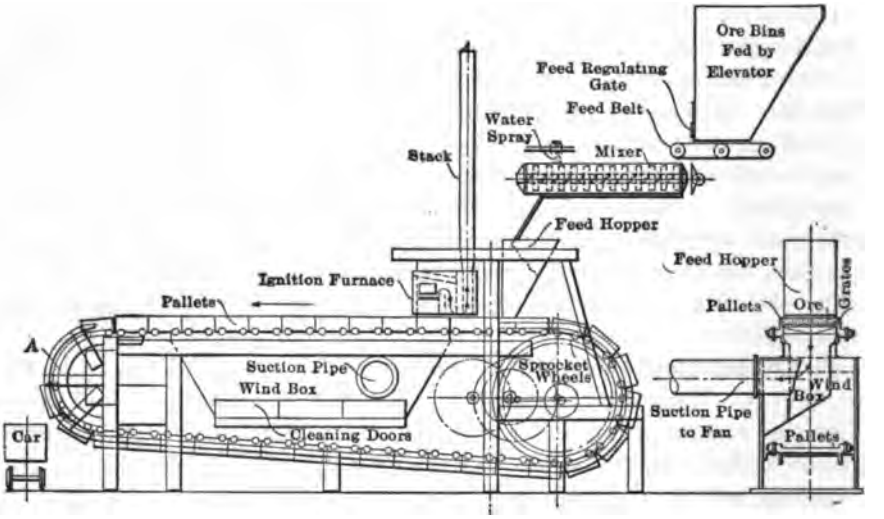
bination, form practically an endless conveyor, with the continuity broken at one place in the circuit. Each pallet is provided with four wheels which engage with the tracks or guides at all parts of the circuit, except where the pallet is passing over the suction-box, and then the pallet slides on its planed bottom over the planed top of the suction-box, thus making an air-tight joint.[1] A pair of cast-steel sprocket-wheels, turning inside of the concentric guide-rails, lifts the train of pallets from the lower to the upper track by engaging their teeth with the roller-wheels, and launches each pallet in a horizontal path under the feed-hopper and igniting furnace, and over the suction-box. In a train of pallets in action, all the joints are kept closed and air-tight by the pallets being pushed from behind. At the beginning and end of the track formed by the planed top of the suction-box, there is a planed dead-plate over which the pallets must glide; which serves to prevent any



FIGS. 152 and 153.—Dwight-Lloyd straight-line machine.

leakage of air. After a pallet passes over the suction-box and terminal dead-plate, its wheels engage the ends of the circular discharge-guides. These are adjusted with the view of raising the pallet about 0.5 in. vertically and thus automatically prying up the cake of sinter and freeing it from the grate-slots. A "breaking-roller" prevents the prying action from extending too far back, and tends to form a line of fracture. This roller, however, is not essential in all cases. On reaching the curves of the guides, the pallets, one by one, drop into the guides, each strikes the pallet which has preceded it and, at the same time, discharges its load of sinter, and shakes free the slots of the grates. The force of the blow can be regulated by the gap left in the train of pallets at this point. The weight of the train keeps the pallets fed down to the lower teeth of the sprocket-wheels.

[1] Improvement by Laws, U. S. Patent No. 1097592, May 19, 1914.

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The complete cycle of operations is as follows: A pallet, being pushed onward tangentially from the top of the sprocket-wheels, passes under the feed-hopper, where it takes its load in the form of a continuous even layer of charge, say 4 in. thick, passes next under the igniter (a furnace in Fig. 152), where the top surface is kindled, and at the same time comes within the influence of the downward-moving current of air, induced by the suction-draft; this carries the sintering action progressively downward until it reaches the grates. The roast-sintering action is complete, the cake is discharged by the pallet dropping into the discharge-guides, and the pallet crowds its way back to the sprocket-wheels, is slowly raised to the upper tracks, and begins a new cycle.

The machines used formerly were smaller than those in operation at present; thus the first machine erected at Salida, Colo., in November, 1908, was 13 by 150 in. The standard machine found in all plants at present is 42 by 264 in.; it weighs, including grate-bars, 58,000 lb., has two suction- or wind-boxes, 42 in. wide and 11 ft. long, each with a suction-pipe, 20 in. in diameter, ending in a suction main 24 in. in diameter, connected with a double-inlet suction fan. The machine including feed-hopper is 40 ft. long, and the tops of the pallets are 21 ft. above the floor. The weight of a pallet including grate is 800 lb. The power required to drive the machine is from 2 to 3 h.p.; that for operating a machine with all accessories except the fan is from 10 to 15 h.p.; the fan proper consumes from 40 to 50 h.p. according to local conditions. The total power consumption per machine is therefore from 50 to 65 h.p. Machines are spaced at 18-ft. centers. A machine with accessories costs $6000, shipping and erecting in an existing building costs about $6000, a building containing automatic feeders, etc., cost about $8000.

The mixing and feeding of the ore-charge is seen to be accomplished, Fig. 156, by delivering the components into a receiving hopper provided with a gate to regulate the discharge, and with an endless belt (automatic feeder) delivering into a horizontal mixer where the ingredients are mixed, and the water, sprayed from a stand-pipe, is incorporated. The mixer discharges through a chute into the feed-hopper of the sintering machine.

The mixer, while very effective, consumes considerable power, and the blades of the revolving shaft wear out quickly. Both disadvantages have been overcome by the plate-feeder shown in Fig. 157. The ore arrives partly mixed on the belt and is discharged by Robins tripper, *b*, into receiving hopper, *d*, where it receives the necessary amount of water through spray-pipe, *e*. Beneath hopper, *d*, rotates circular feed-table, *f*, provided with stationary pivoted rabbles (not shown, similar to those of the Brunton or Godfrey furnace, Figs. 131–132 and 133–137) for mixing the ore and moving it from center to periphery, and a pivoted scraper (not shown) to discharge the mixed and moistened ore into feed-hopper, *g*, whence it drops on to inclined plate, *h*, and forms ore-bed, *i*, on pallets, *j*, traveling in the direction of the arrow. Hand-chain, *k*, serves to change the direction of the pivoted rabbles and to adjust the feeding. In forming ore-bed, *i*, the coarser particles roll down the inclined surface on to the

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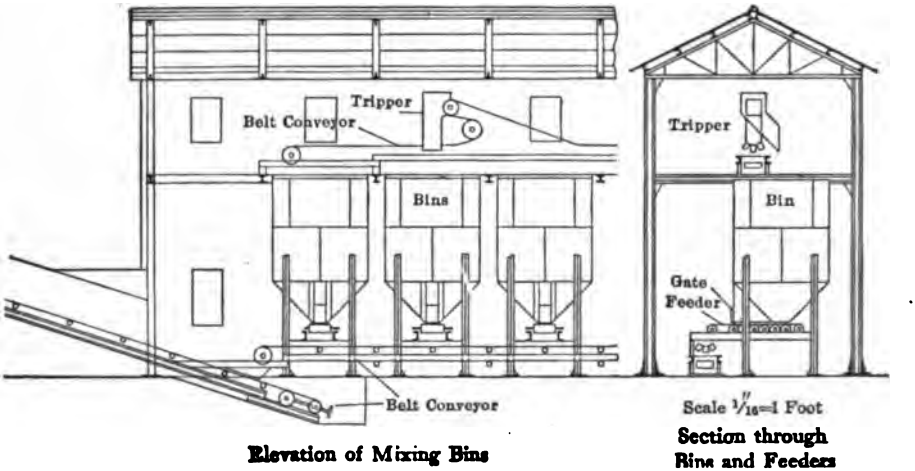
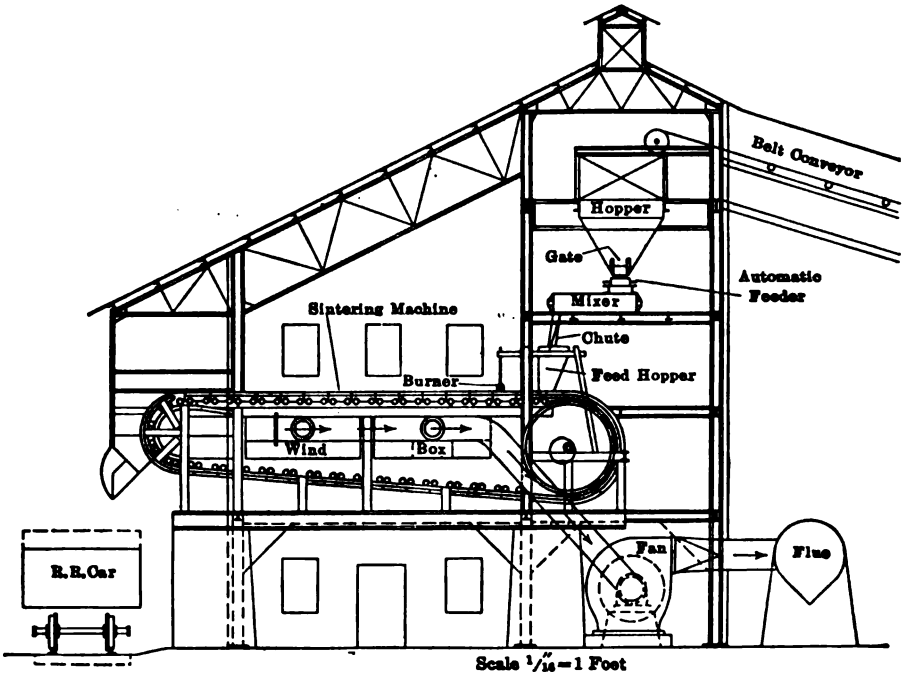
pallets and form a porous bed, which is essential for good work. As the pallets travel under the igniter, the finer particles at the top of the triangular ore-heap are pushed back over the coarser by the vertical side of the feed-hopper. This





FIGS. 154 to 156.—Plant of Dwight-Lloyd straight-line machine.

terminates 4 in. above the pallets and thus furnishes the desired uniform layer of ore, which in the standard machine is 42 in. wide and 4 in. thick.

Before this manner of distributing ore-mixture on the grate was devised,

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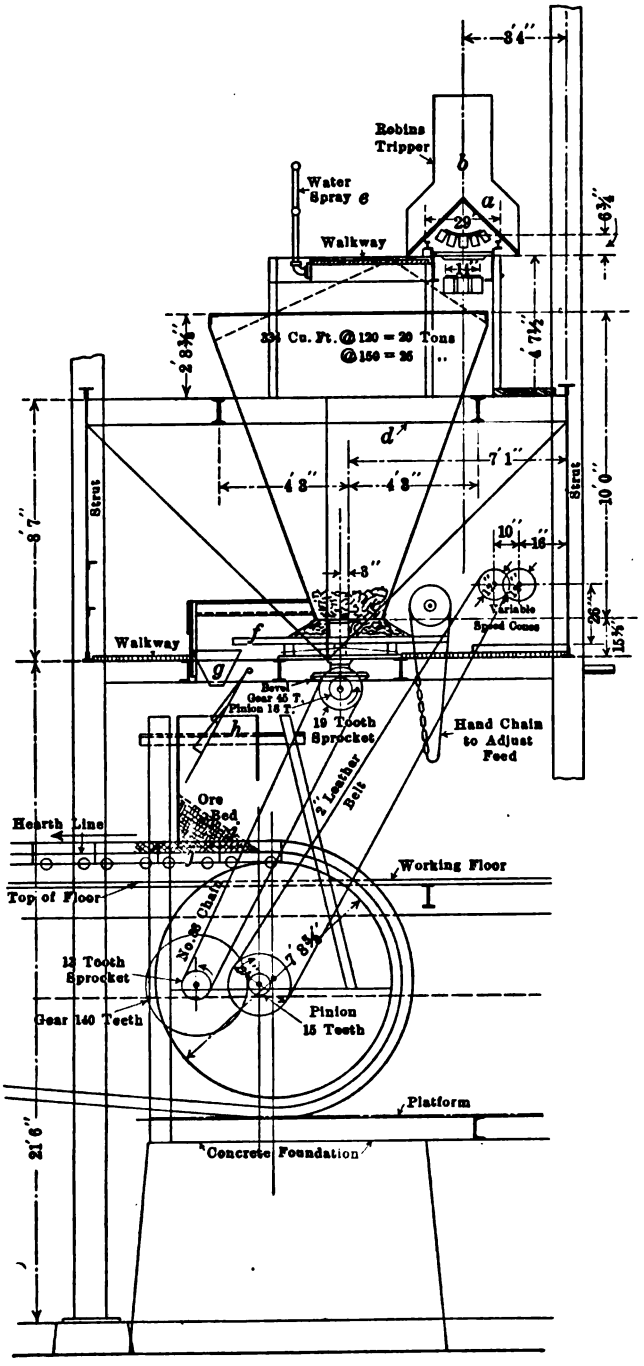

FIG. 157.—Plate-feeder for Dwight-Lloyd straight line machine.

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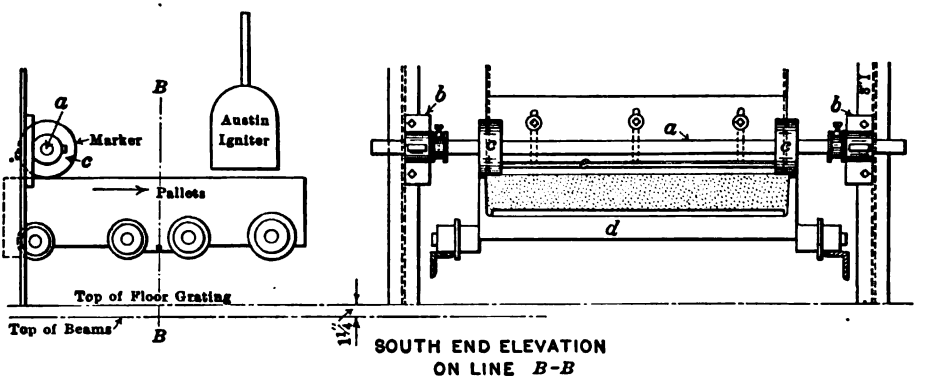
machines frequently had two feed-hoppers, one back of the other, the hopper farther away from the suction-box receiving coarse ore-mixture or crushed limestone, the other fine material. The two hoppers are still in place with older machines, but only one is at present in operation.

When the sinter-cake has passed the suction-box and a piece is broken off to be discharged, the break is often rather jagged. In order to obtain a straight-line break, a so-called "Marker" has been introduced at one plant between the feed and igniter which makes at the end of a pallet an indentation on the ore-surface and causes the break to form along a straight line. This simple and effective device is shown in Figs. 158 and 159. Horizontal shaft, *a*, revolving in boxes, *b*, attached to 8-in. I-beams, carries two rollers, *c*, driven by traveling pallets; rollers, *c*, are joined by flexible iron bar, *e*, 1.5 by 1.25 in., which makes a cut into the ore-bed with every revolution of the shaft.



FIGS. 158 and 159.—Marker for Dwight-Lloyd sintering machine.

The ore-charge was originally, and in some instances still is, ignited from a small suspended coal-burning furnace built of tiles, Figs. 152 and 153. The grate has the same width as the pallet; it is about 15 in. wide, and burns about 1,000 lb. coal in 24 hr. The flame, after passing over the firebridge, is deflected downward upon the ore by a brick curtain which can be raised and lowered, and is then drawn upward by the natural draft of a small stack.

Gasoline and similar light hydrocarbons have been used to kindle the surface of the ore, but have been abandoned on account of the high prices. They have been replaced by residuum oil of about 26° Bé. At present most machines use an oil-burner; of the many suggested, the J. F. Austin burner or igniter[1] is found almost everywhere. It is shown in Figs. 160–163. In Figs. 160 and 161, *A* represents a steel frame of the Dwight-Lloyd machine, *B* the feed-hopper, *C* the pallet, *D* the sprocket wheel, *F* the ore-charge, *G* the igniter, and *H* the suction-box. Igniter *G* consists of an arched refractory firing chamber suspended by adjustable rods in front of feed-hopper, *B*. The chamber, open along the larger part of the bottom, is formed of two kinds of arched brick enclosed by an iron casing; the side-bricks, Fig. 163, have no bottom, but the two end-

[1] U. S. Patent No. 1179390, April 18, 1916; *Min. Eng. World*, 1916, XLV, 136.

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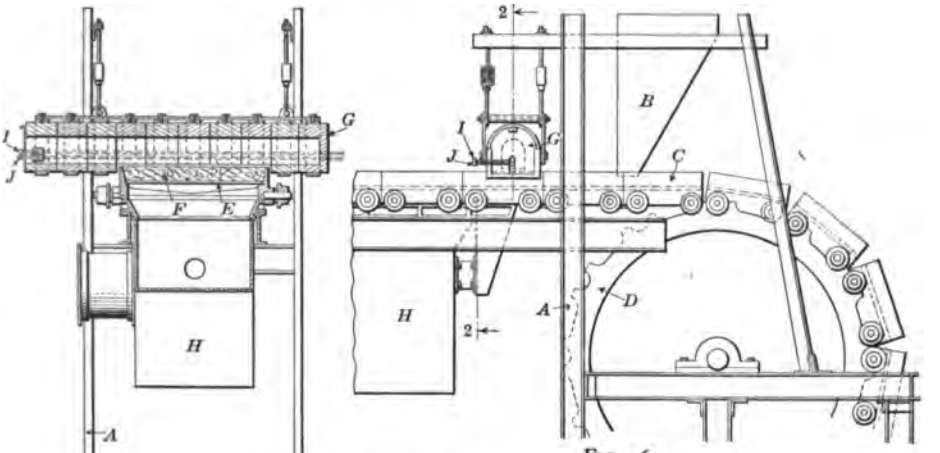


FIG. 160.

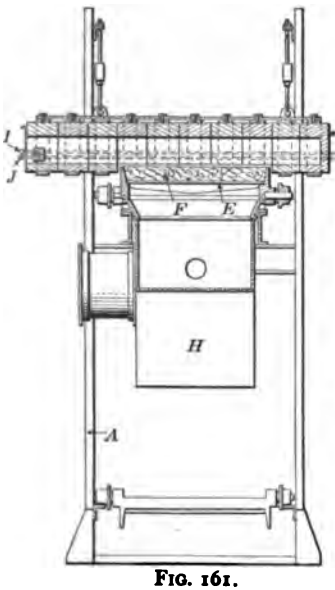


FIG. 161.

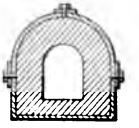


FIG. 162.

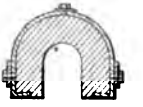
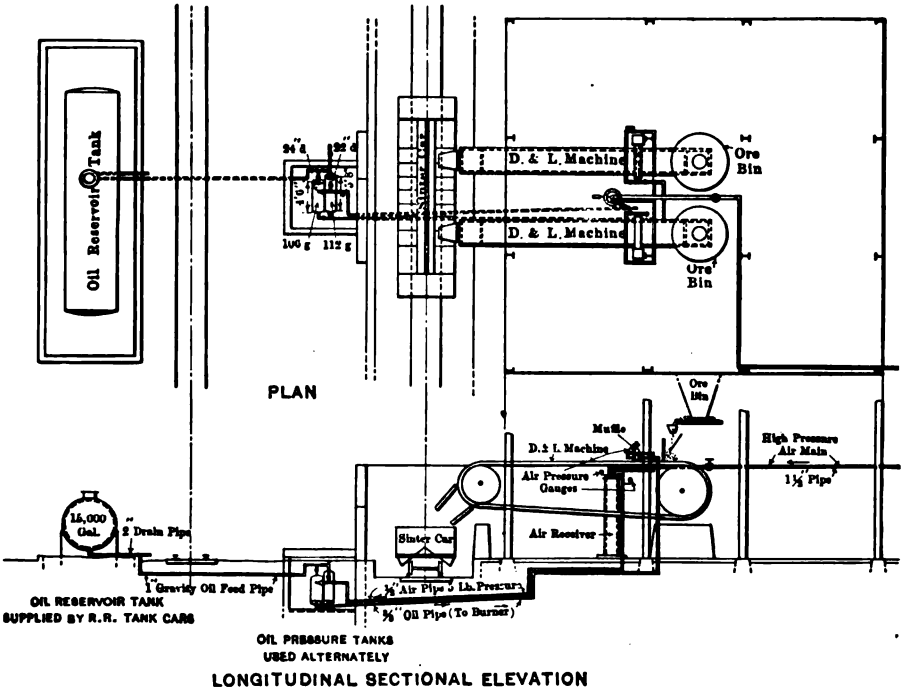


FIG. 163.

FIGS. 160 to 163.—J. F. Austin oil igniter.



FIGS. 164 and 165.—Plan and longitudinal elevation of oil- and air-feeds to Austin igniters.

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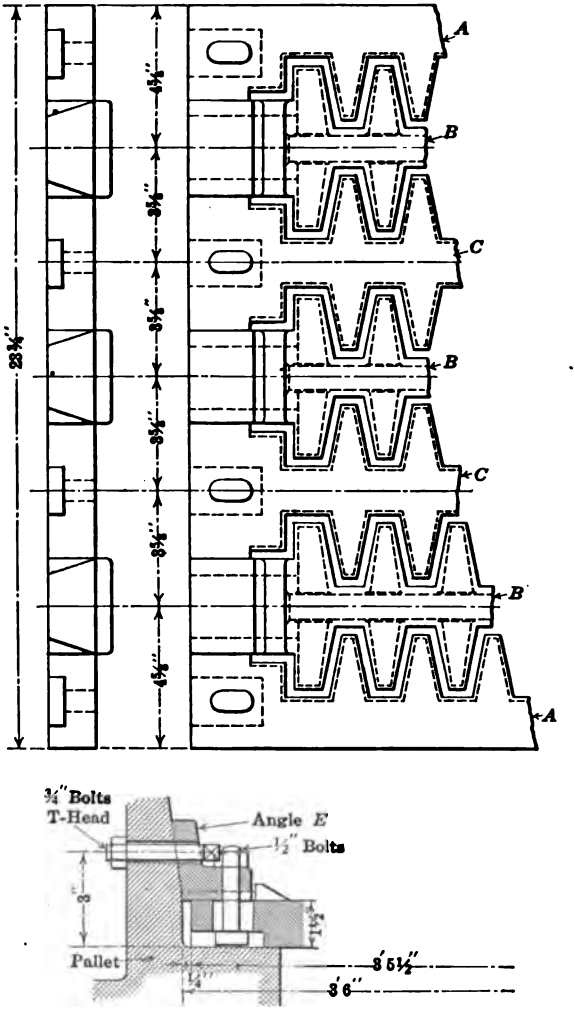
bricks, Fig. 162, have bottoms and openings through which pass the oil- and air-pipes, *I* and *J*. The oil flame enters the combustion chamber and fills it with flame. As the chamber extends on either side beyond the pallets, the whole surface of the charge is uniformly heated as it passes beneath the chamber. The downdraft of suction-box, *H*, draws downward the flame and hot gases.



FIGS. 166 to 168.—Stewart grate for Dwight-Lloyd machine.

The general arrangement of oil- and air-supply for two Dwight-Lloyd machines with Austin ingniters is shown in plan and longitudinal section in Figs. 164 and 165, which are clear without further comment. The air-pressure is given as being 5 lb. per sq. in.; in some plants the oil-pressure reaches 20–25 lb., and the air-pressure as much as 40 lb. per sq. in.

The pallets, Figs. 152 and 153, usually are cast-iron frames and have herring-bone grates. In some instances the frame is a malleable casting, and the grate

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has straight slots. Malleable iron stands shock better than an ordinary casting and straight slots are more easily freed from adhering clinker than the diagonal. Usually one man is required to remove adhering clinker with a chisel-pointed bar about 4 ft. long; in some instances a hammer-drill is used to do the work more quickly. The same man closes up, *e.g.*, with a piece of paper, any slight breaks in a grate until this is worn out sufficiently to warrant replacing. Mechanical cleaners[1] have been tried, but have not been satisfactory, as the striking disks do not alway fit the slots and hit the ribs. The Stewart grate in operation at Trail, B. C., shown in Figs. 166–168, obviates the necessity of punching. The grate, 42 by 23¾ in., consists of fixed end- and center-grates, *A* and *C*, fastened to the pallets, and swinging grates, *B*, pivoted in the ends of the pallets. Thus, when a pallet passes over the discharge-end of the machine, the swinging pallets, *B*, are slightly tilted and break any clinker which may be present. Clinkers, however, do not form always, as with suitable ores and other favorable conditions, sintering machines are in operation which dispense entirely with cleaning of grates.

The suction-box in the original small-size machine was 30 by 102 in. at the top, giving an effective grate area of 31.25 sq. ft., which is the true hearth area of the machine. The present standard machine, 42 by 264 in., has two suction-boxes, 48 by 132 in., representing a grate area of 38.5 sq. ft. In most modern machines the suction-boxes are providedwith a screw-conveyor along the bottom which, set going at intervals, removes the collected dust into a vertical discharge-pipe with two gates, one near the top the other near the bottom. By this arrangement the dust can be removed without temporary loss of suction.

The sinter-cake containing a small percentage of fines used to be and still is discharged in most cases, Figs. 152 and 153, through a chute into a railroad car. With a high speed of pallets there is produced a larger amount of fines than when the pallets travel slowly. At Tooele, Utah, W. Wraith dumps the cakes over an inclined grizzly with ½-in. slots, separates the high-S fines from the low-S cakes, and returns the former to the ore-charge. By this addition of half-treated ore, he loosens up the charge and diminishes the formation of clinker to such a degree that cleaning of grates has become the exception. His example is being followed by other plants.

In dumping a product containing many fines there is formed much dust. In order to collect this dust, the receiver for the fines is covered with a hood which ends in a rectangular inverted trough covering a large part of the roasting bed; it extends, *e.g.*, to 6 ft. from the burner. The downdraft on the bed sucks the dust downward upon and into the roasting charge where it is more or less agglomerated. The hood at the same time conserves heat and reduces the temperature of the floor of the building.

The leading facts regarding the operation of the Dwight-Lloyd straight-line sintering machine are given in Table 47.

[1] Editor, *Eng. Min. J.*, 1913, XCVI, 989.
Newhouse, *Tr. A. I. M. E.*, 1914, XLIX, 527.

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TABLE 47.—STANDARD DWIGHT-LLOYD STRAIGHT-LINE SINTERING. MACHINE, 42×264 IN.

	Tooele, Utah	Salida, Colo.	Herculaneum, Mo.	Midvale, Utah	A. S. & R. Co.	A. S. & R. Co.	A. S. & R. Co.	Northport S. & R. Co. Northport Wash.	Trail, B. C.
PALLET, SPEED OF TRAVEL, in. per min.	26–30	7.5–22.5	16	24	16.5–51	10–33	18–28	6–8	26
SUCTION-BOX, VACUUM, in., water	5	5.5–8.0	4–8	10.5	.....	0.4–0.7	6–9	.....	3
IGNITER-FUEL KIND.....	Oil, 28–32° Bé.	Lump coal	Oil, 25–26° Bé	Oil 24–28° Bé	Oil	Oil, 35° Bé.	Oil, 39° Bé.	.....	Oil, 28° Bé.
Amount per ton charge	1 gal.–2.257	1 ton–204.4	1 gal.–0.695	1 gal.–2.500	1 gal.–2.5 to 3.3	.....	1 gal.–3.57	.....	1 gal.–1.000
CHARGE MIXTURE:									
Character.....	(1)	(6)	(10)	Variable	(13)	(16)	(20)		(26)
Size of particles.....	(2)	0.25 in.	⅜ in.–200-mesh	½ in.–200-mesh	⅜ in. to dust	0.355	5/16 in. to fines	¼-⅜in.	Ore ¼ in., limestone ¾ in.
Chemical analysis.....	(3)	(7)	(11)	Variable	(14)	(17)	(21)	(23)	(27)
Water, %.....	8.6	7–10	5+	8–10	5–8	3–5	8–10	8–10	.....
Depth of bed on grate, in.	4.5	4.5	4	4.5	4	4	4	3–4.5	.....
Distribution of coarse and fine on grate	(4)	(8)	Coarse bottom	Coarse settles to bott'm	Not considered	Coarse on bottom	Coarse on bottom	(24)	Coarse on bottom
Use of limestone bed...	Not used	Not used	Not used	Not used	Not used	Not used	Not used	Not used	Coarse limestone settles on bottom
Tons in 24 hr.....	200–225 incl. 160 excl. } return material	92 new material	80–120	100–150	388 (18)	150–250(18)	200 ?	72–120	100
BLAST-ROASTED CHARGE:									
Loss in weight, %	15.28	12.5	5	N. d.	4–5	5	8	.....	N. d.
Fines, %	19.1	15.0	5	Variable	Not separated	10–15	15	7	N. d.
Sulphide-S, sulphate-S, %	Total S 3.07	5.0–0.6	3.5–1.0	N. d.	8.4–0.5	4.5–1.5	4.2–0.4	N. d.	Total 4–5
FLUE-DUST:									
Amount, % of charge.....		0.2	0.5	N. d.	N. d.	0.3	0.8	N. d.	1.5 (18)
Chemical analysis:....	(5)	(9)	(12)	N. d.	(15)	(19)	(22)	(25)	(28)
GAS:									
Volume, cu. ft., 20° C., 760 mm.	9,200	10,200	9,000	6,700	9,500–11,200	12,000–15,000	12,000	N. d.	10,000
% volume, SO_2, SO_3.....			2.9–0.001	1.5–low	N. d.	?–0.0032	3.2–0.000101	N. d.	1.2–none
LABOR:									
Tons per man per day	11.36	10.82	16.66	22.72	43.21	10.53	44.44	29.00	22.22

(1) Ore 33.6, concentrate 12.37, slime 2.75, iron ore 25.55, slag 0.01, calcine 23.6, fume and flue-dust 2.22 per cent. (2) +2–4.9, +5–15, +10–11.5, +20–12.6, +30–10.1, +60–15, +100–13.3, –100-mesh–17.6 per cent. (3) Pb 15.25, Cu 0.83, Fe(Mn) 24.6, S 11.5, CaO 3.8, Zn 5.8 per cent., Ag 11.27, Au 0.04 oz. (4) Coarse on bottom grading to fine on top. (5) Cu 0.25, Pb 37.8, Insol. 2.5, Fe 4.6, S 29.7, CaO 0.7 per cent.; Ag 5.5, Au 0.01 oz. (6) Lead concentrate, pre-roasted iron sulphide, silica and silicious sulphide; pre-roasted iron sulphide, silica and silicious sulphide; pre-roasted matte, silica and silicious sulphide. (7) Pb 16.2, Cu 2.8, Insol. 21.4, Fe 25.3, Mn 1.0, Zn 6.5, S 14.1. (8) Coarse on bottom fine on top. (9) Cu 0.7, Pb 22.0, Insol. 9.8, Fe 11.9, S 34.4 (including free S) Zn 6.1. (10) Wet galena concentrate, sand, pre-roasted matte, pyrite, cinders. (11) SiO_2 14, Fe 23, CaO 2.5, S 11, Pb 43. (12) Pb 69, Zn 1.0, S 11.3. (13) Lead concentrate flotation slime, fume, raw matte. (14) Pb 47.5, SiO_2 8.6, FeO 13.3, CaO 3.7, MgO 1.9, Al_2O_3 2.5, Zn 3.5 S 12.5. (15) Pb 43.0, SiO_2 11.0, FeO 6.6, CaO 4.5, MgO 2.4, S 11.4, Al_2O_3 5.4, Zn 4.2. (16) Galena, flotation slime, silicious sulphide, limestone. (17) SiO_2 6, Fe 18, CaO + MgO 2–10, S 12, Zn 5.5, Pb 35–41, Cu 0.5. (18) Double roasting. (19) SiO_2 6, Fe 9, CuO 1.5, S 2.5, Zn 3.5, Cu trace, Ag 18 oz., Au trace. (20) Washed lead ore, sulphides, matte, silicious oxide lead ore. (21) Pb 20, Cu 2, SiO_2 19, Zn 9.6, Fe 18, S 13, CaO 4. (22) Pb 27.6, Cu 1.1, SiO_2 12.9, Zn 0.3, S 19.2, Fe 12.2. (23) SiO_2 12–16, Fe 16–18, CaO 2–3, S 8–14, Pb 34–40. (24) Coarse silicious ore and limestone on bottom. (25) Pb 50, Fe 3.9, SiO_2 9.8, S 11.3. (26) Roasted lead ore, matte, limestone. (27) Pb 25, Fe 7.8, SiO_2 15, CaO 8, S 10–11, Zn 8.9. (28) Pb 28.8, Fe 17.5, SiO_2 9.0, CaO 8.0, S 11.0, Zn 9.6 ($PbO : PbSO_4 = 6.4 : 1$).

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The speed of travel of the pallets which ranged from 7 to 10 in. and reached perhaps 15 in. per min., has been advanced to as much as 30 in. with a corresponding increase in tonnage from 50 to 200 tons and over. This has been accompanied by a rise in the S-content of the sinter which, however, did not correspond, as might be expected, to the quantity of ore treated. At the Tooele plant[1] the speed of travel of pallets was 13 in. per min., and the daily product 70–95 tons; a slight advance in speed gave an inferior product with high S-content; an increase to 26 and 30 in. more than doubled the product which had a good structure and a low S-content. From this it appears that in increasing the speed of travel one reaches a halting-point; but that as soon as one gets well beyond it, new conditions arise which permit a quicker travel. An explanation may be found in the fact that the charges being fed quickly do not pack, pass quickly under the igniter and do not matte, and can stand more water than with a slow travel; the greater the permissible water, the more porous the charge, and the more quick the roast. During 5 months the average analysis of the ore mixture was Pb 13.51, Cu 0.694, SiO_2 22.46, Fe-Mn 25.50, S 11.27, CaO 3.99, Zn 4.93 per cent., Ag 10.169 and Au 0.0472 oz. per ton; that of the product: Pb 14.56, Cu 0.92, SiO_2 23.92, Fe-Mn 28.63, S 2.78, CaO 4.28, Zn 5.85 per cent., Ag 11.23 and Au 0.052 oz. per ton. A screen analysis of the ore treated in 4 weeks gives the data shown in Table 48.

TABLE 48.—SCREEN ANALYSIS OF ORE MIXTURE, TOOELE, UTAH

Size, mesh	December, 1915			
	7	14	21	28
	Per cent.	Per cent.	Per cent.	Per cent.
+ 2	5.0	2.5	7.4	4.7
+ 5	15.9	11.6	16.6	15.8
+ 10	11.7	11.3	11.7	11.2
+ 20	12.4	12.7	12.4	13.0
+ 30	10.0	10.5	9.3	10.7
+ 40	12.4	3.9	3.7	3.5
+ 60		11.6	11.7	13.4
+100	11.7	16.5	11.7	13.4
−100	20.9	19.4	15.5	14.3

The amount of grizzly fines varied from 15 to 20 per cent. A screen and a chemical analysis are given in Table 49. The screenings are seen to assay 5–6 per cent. S, whereas the caked product contains 2–3 per cent. S. Returning the grizzly screenings to the ore-charge makes the latter porous and facilitates the roast. There is thus obtained with a large tonnage for ore, a double treatment of the high-S product.

[1] Private communication by W. Wraith, February, 1916, and private notes, July, 1916.

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TABLE 49.—SCREEN AND CHEMICAL ANALYSES OF GRIZZLY FINES, TOOELE, UTAH

Size, inch	Per cent.	S	Pb	Zn
−½	100.	6.3	12.4	8.5
+⅜	9.1	6.7	11.0	9.0
+5/16	9.9	4.9	10.1	8.0
+¼	14.4	4.2	9.8	7.2
−¼	66.6	6.4	13.8	8.6

The average loss in Pb with high-speed travel of pallets is from 1.5 to 2 per cent. A few data covering losses are given in Table 50.

TABLE 50.—LOSS OF METAL IN HIGH-SPEED DWIGHT-LLOYD MACHINE (TOOELE)

Tonnage	Loss, per cent.			
	Pb	Cu	Ag	Au
147	0.79	0.60	0.913	0.137
115	0.66	0.85	0.273	0.173
110	1.30	0.17	0.663	0.409
127	1.20	0.79	0.303	0.950
124	0.72	0.94+	0.950	1.387+
116	4.68	.....	4.380	.....

+ denotes an apparent gain.

The vacuum in the suction-box, which is governed largely by the density of the charge, shows a range of from 5 to 10 oz.

Ignition fuel has been taken up on page 196. The amount of oil required per ton of charge ranges usually from 2.5 to 3.0 gal.

The charge is made up of raw and pre-roasted sulphides, matte, silicious ores, and sometimes limestone. The use of pre-roasted pyrite concentrate as iron flux often shows a tendency to hinder the agglomeration of particles. An addition of matte usually corrects this evil. The size of charge components shows a range from ½-in. to 200-mesh. It is essential that the ingredients of a charge be well mixed and uniformly moistened. The S-content varies between 11 and 14 per cent. The amount of Pb covers a large range extending from 15 to 47, and even 50 per cent. The percentage of SiO_2 is 10 to 25, that of FeO is about $1.5 \times SiO_2$; CaO is usually lower than FeO.

In making up a charge, the size of particles deserves about as much consideration as the chemical composition, as it is of vital importance that the charge be porous. The quantity of water added varies with the calorific power of the charge and the percentage of fines; the usual range is from 5 to 10 per cent.; the finer the charge-components the greater has to be the water-content. The depth of charge is uniformly 4 in. With the von Schlippenbach table-machine (§97) the depth is 10 in., for the purpose of obtaining

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a gas richer in SO_2 than is feasible with a 4-in. bed; but the greater the depth, the higher the S-content and the density of the sinter.

Feeding of charge has been discussed on page 193.

The loss in weight varies greatly with the charge as does the amount of non-sintered fine material. The analyses in Table 49 show that the sinter retains from 3 to 4 per cent. S, and this is mainly sulphide-S. The amount of unsintered fine material, given in Table 49, is large, 10 to 15 per cent.; this is due in part to the fact that flotation concentrates form a considerable part of the charge. With table concentrates the fines do not exceed 5 per cent. unless the speed of the pallets is high. The amount of flue-dust formed is small, 1 per cent. and under. If pyrite forms a large part of the charge, the molecule of S which passes off at a low temperature is found in the dust in the form of flower of sulphur. Such a mixture is likely to be self-igniting,[1] cause an explosion, and damage the bag house. Accidents of this character are avoided by subjecting pyrite to a rough roast before it is added to the charge, or, if this is not feasible, by keeping pyrometers in the flues, and allowing the gases to pass into the open, should the temperature rise above the normal of 150 to 200° C.

The volume of gas escaping from the furnace is about 10,000 cu. ft. per min.; it contains from 1 to 3 per cent. vol. SO_2, and only traces of SO_3.

In blast roasting at Cerro de Pasco, Peru, 14,000 ft. above the level of the sea, Lloyd[2] found that the igniting flame had to be hotter, that the charge could stand a higher S-content than usual, and that the speed of roasting was slightly less than at lower elevations; he also noticed that flower of sulphur which collected in the fan did not ignite of its own accord.

99. General Arrangement of Dwight-Lloyd Plant.—The general arrangement has taken a standard form of which diagrammatic sketches are given in Figs. 154–156.[3] The ores to be blast roasted arrive on a belt-conveyor at the top floor of the building containing the mixing bins and are discharged by means of a tripper into cylindrical hopper-bottom bins. The content of each of these bins is discharged in the desired amount by its traveling belt through a regulating gate on to a main belt-conveyor which delivers on to an inclined belt conveyor raising the unmixed charge to the feed-hopper at the top of the roaster building. This holds the charge, now mixed somewhat, but not sufficiently to furnish a uniform product. This uniformity is obtained in the mixer.

At Tooele, 20 cylindrical ore-bins with gates and Challenge feeders deliver the contents on to the traveling belt.

The roast-sintering plant of the Bunker Hill Smeltery, Kellogg, Idaho, is shown in Figs. 169–171.[4] It contains one Wedge multiple-hearth furnace,

[1] Editor, *Met. Chem. Eng.*, 1913, XI, 426.
Anderson, *Tr. A. I. M. E.*, 1914, XLIX, 577.

[2] *Min. Sc. Press*, 1913, CVI, 908.

[3] Drawings of plant of Ohio and Colorado Smelting & Refining Co., Salida, Colo., have been published in *Met. Chem. Eng.*, 1912, X, 87.

[4] Dwight, *Eng. Min. J.*, 1916, CII, 674.

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21 ft. 6 in. in diameter and 22 ft. 11½ in. high, and four standard Dwight-Lloyd machines treating each about 200 tons in 24 hr. Details of the Wedge furnace have been given in connection with Figs. 139 and 140; those of the Dwight-Lloyd machine have just been taken up. The general arrangement of

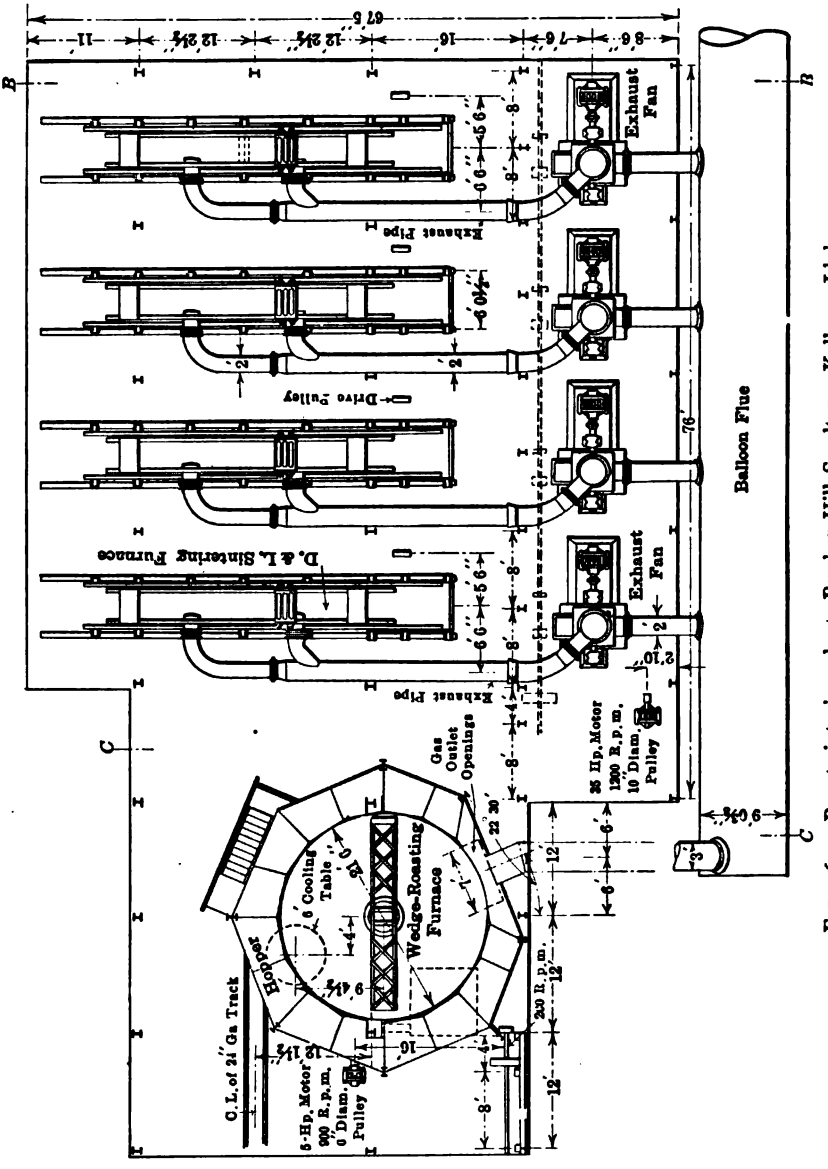

FIG. 169.—Roast-sintering plant, Bunker Hill Smeltery, Kellogg, Idaho.

plant is given with much detail in the drawings and needs no further comment in view of what has already been said.

100. Double Treatment.—By this term are understood the operations of rough-roasting a large amount of ore in a Dwight-Lloyd machine, and finishing

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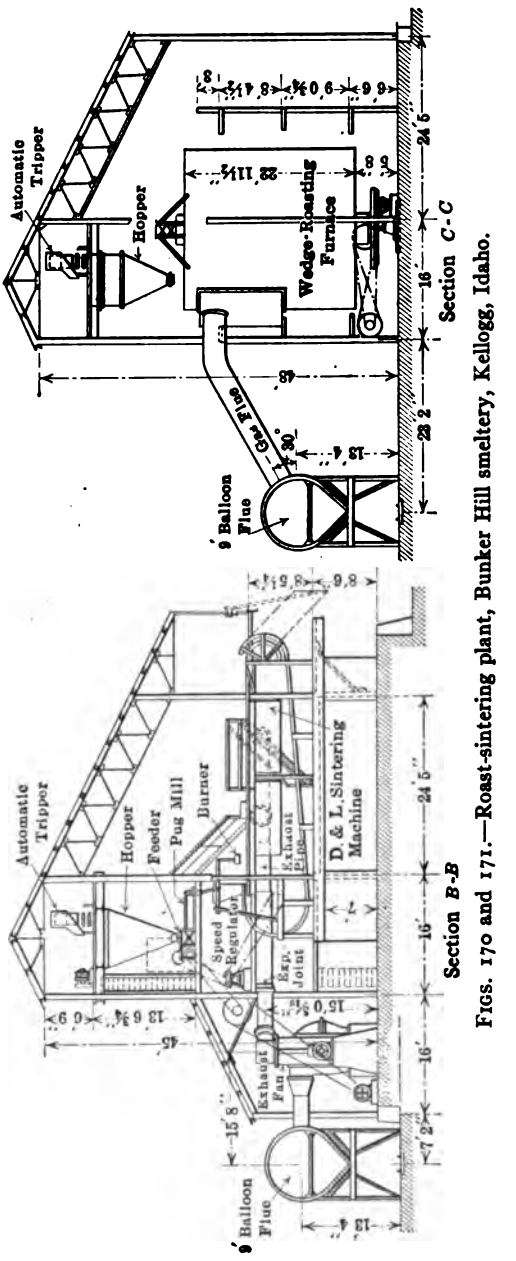
the rough-roasted material either in a Huntington-Heberlein pot or in a second Dwight-Lloyd machine in order to obtain a porous sintered product low in S-content.

A combination of D.-L. and H. & H. treatment has been worked out satisfactorily[1] in two plants.[2] The D.-L. mixture contains, *e.g.*, SiO_2 9, Fe(Mn)O 16, Ca(Mg)O 8.5, S 10–11.5, Zn 6.5–7.5, Pb 28–32 per cent., Cu trace. The pallets are moved at the high speed of 37–48 in. per min., so that about 250 tons charge are put through in 24 hr.; the S-content of the charge, which may go as high as 15 per cent., is reduced to 6–7 per cent. The product is broken in a Williams hinged-hammer crusher to pass a ⅜-in. screen, then mixed with 10–12 per cent. limestone crushed to the same size, and blown in an H. & H. pot.

A charge of 12 tons mixture furnishes in 5–6 hr. 9–9.5 tons sintered product with 1.5–2.5 per cent. S, of which 40 per cent. is sulphate-S. The pot is given first a 2-in. grate-dressing of silicious ore, then a 1-in. layer of glowing coal from a Godfrey furnace, and finally its full charge to be blown at a pressure of 12–14 oz. The advantages of the double treatment are that the double roast furnishes more ZnO than does a normal single roast, that the matte-fall is greatly reduced and with it the danger of zinc-mush, and that the direct output of lead is largely increased. These advantages more than balance the cost of the additional operation.



FIGS. 170 and 171.—Roast-sintering plant, Bunker Hill smeltery, Kellogg, Idaho.

[1] Riddell, *Tr. A. I. M. E.*, 1914, XLIX, 531.

[2] Hofman, private notes, July, 1916.

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In the combination of two D.-L. machines the mode of operating is similar to the double treatment just described. It was started by H. L. Hyde in Australia about 1911. The original charge, crushed to ½-in. size, containing, *e.g.*, SiO_2 8–10, FeO 28–30, CaO 4–5, S 12–13 per cent., is run on the machine with pallets traveling at a speed of 30 in. per min. The rough-roasted partly caked material retains 7–8 per cent. S, is crushed in a Jeffrey swing-hammer machine to pass a ⅜-in. screen, and is retreated without the addition of limestone, the pallets traveling at a rate of 22 in. per min. The finished product, which is imperfectly sintered but practically free from fines, retains about 2 per cent. S.

The practice inaugurated by Wraith at Tooele (page 201) of having a quick travel of pallets, and of separating high-S fines from low-S sinter by means of a grizzly, is actually a double treatment, except that only high-S fines, which form 15–20 per cent. of the original charge, go to be treated again. The advantage of this method of working is evident, especially with ores that contain little blende.

101. Comparison of Huntington-Heberlein and Dwight-Lloyd Processes.—The general comparison of intermittent up-draft and continuous down-draft blast roasting, given in §97, is here supplemented by the work of Norton[1] at Murray, Utah, and of Riddell[2] at East Helena, Mont. The former, on the whole, favors the H. & H., the latter the D.-L. process. A summary of the conclusions arrived at by the two superintendents of these prominent plants is given in Table 51.

TABLE 51.—COMPARISON OF HUNTINGTON-HEBERLEIN AND DWIGHT-LLOYD PROCESSES

Item	Advantage in favor of	
	At Murray	At East Helena
1. Cost of installation	D.-L.	D.-L.
2. Cost of roasting	H. & H.	Even
3. Adaptability to charge	D.-L.	D.-L.
4. Metal loss	Even	Even
5. Physical condition of product	H. & H.	D.-L.

The table shows agreement as regards items 1, 3, and 4. The disputed headings 2 and 5 find, in a general way, their explanation in the fact that wherever a new installation is made, the D.-L. is preferred to the H. & H., and that the slightly greater cost of the H. & H. in a large plant is not of sufficient importance to explain the preference shown to the D.-L.

In comparing the behavior of the H. & H. and the D.-L. products in the blast-furnace, Norton[3] found that in smelting 8,000 tons charge with 950 tons or 11.5 per cent. coke, he obtained with the H. & H. material slags assaying 0.81 per cent. Pb, while with D.-L. material they assayed 1.03 per cent. He

[1] *Tr. A. I. M. E.*, 1914, XLIX, 485.
[2] *Op. cit.*, 495.
[3] *Op. cit.*, 490.

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argues therefrom that the reduction with the product from the H. & H. is better than with that from the D.-L.; or that with the latter you require more coke to obtain the same result as with the former.

Dwight[1] points out that in other plants less coke was required with the D.-L. than with the H. & H. product, and gives as reason for this that in the former iron is present to a large extent as FeO, while with the H. & H. the compound Fe_2O_3 is more prevalent. He gives in Table 52 the smelting power of a blast-furnace, 48 by 162 in. at tuyères (= 54 sq. ft.) with charge-materials that have been subjected to different kinds of preliminary treatment.

TABLE 52.—SMELTING POWER OF BLAST-FURNACE, 42×162 IN. = 54 SQ. FT., WITH MATERIALS ROASTED AND BLAST ROASTED

Ore	Tons	
	In 24 hr.	Per sq. ft. hearth area in 24 hr.
Roasted in Brückner cylinder.....	180	3·7
Blast roasted in H. & H. pot.....	200	3·7
Screened, and coarse blast roasted in H. & H. pot.	227	4·2
Screened, and coarse blast roasted in D.-L. machine.	285	5·3

The data bring out the increased tonnage with blast roasted product over ore which has been only roasted; they show that the porous sinter of the D.-L. product is more readily smelted than the denser product of the H. & H. process. It is generally acknowledged that the D.-L. machine furnishes a more porous product than does the H. & H. pot. This is especially the case with charges rich in Pb as at East Helena, Mont., where according to Newhouse[2] 60 per cent. of the H. & H. cake is slagged. A porous material absorbs heat more readily than one that is dense, a sintered product is more readily reduced than ore that is slagged; both considerations point to a smaller coke consumption and a greater tonnage with sintered than with slagged material on the charge.

102. Blast Furnaces, Handling of Raw Materials and Products, General.—The blast furnaces in operation at modern plants resemble one another; in fact they have assumed standard shapes. In the handling of materials and products there is less uniformity, as hand labor is being gradually replaced by machinery. Improvements in both cases are the result of the competition of centrally located silver-lead smelteries of the United States to meet existing requirements.

The first furnaces were small brick structures, 20 by 30 in. at tuyères; they had only short campaigns. In 1871[3] the Piltz polygonal and the Rasch-

[1] *Op. cit.*, 492.

[2] *Tr. A. I. M. E.*, 1914, XLIX, 528.

[3] Hahn-Eilers-Raymond, *Tr. A. I. M. E.*, 1871–73, I, 91.
Ingalls, *Eng. Min. J.*, 1907, LXXXIV, 1057.
Ingalls, W. R., "Lead and Zinc in the United States," McGraw-Hill Book Co., New York, 1908, 48.

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ette oblong brick furnaces, common in European plants, were introduced at Eureka, Nev.; ten years later the oblong furnace had superseded the polygonal and circular forms. Cast-iron water-jackets began to replace in 1872 the brick-work of the smelting zone.[1] In 1871 Arents[2] had invented his siphon tap (§110). In vertical section the furnaces at first were prismatic, later the sides were made to taper toward the bottom, and lastly the tapering sides received boshes. The crucible originally partly internal and partly external, was replaced by the internal form with the adoption of the Lürman tapping-jacket. External crucibles have been patented,[3] but are not in use, as far as the writer is aware. Thus about 1880 the leading features, common to all ore-furnaces, were established. The furnaces are oblong in horizontal section, have tapering sides with boshes, straight ends, a water-jacketed smelting-zone, an internal crucible, and an Arents siphon tap. The circular water-jacketed furnace is used only to smelt at intervals small quantities of intermediary products of a refinery. The brick shaft common with most ore furnaces is now giving way to steel water-jackets.

With the increase of size of furnaces and of smelting plants the moving of ore, flux, and fuel by teams and wheelbarrows became impossible and had to be replaced by mechanical traction and manipulation.

The accessory apparatus to remove the slag from the first small-size furnaces treating mainly oxide ore consisted of simple forms of slag-pots. With the advent of sulphide ore provision had to be made to collect matte. Matte-settling pots used at first have given way to oblong fore-hearths. Finally, the disposal of the increasing amounts of intermediary and final products necessitated the introduction of steam, compressed-air, and electric power, so that at present mechanical traction and manipulation has become typical.

Formerly, no provision was made to save the values contained in the waste gases; later, dust chambers were provided, and these have been supplemented by filter-bag houses and electric precipitating apparatus.

The primitive blowing machines used at first have been scrapped; rotary pressure blowers of the Roots type have been installed, and their supremacy is now being challenged by turbo blowers.

Blast-furnace, General.[4]—Five blast-furnaces in operation at some of the leading smelteries are represented in Figs. 172–185. Figs. 172 and 173 show the standard furnace built by the Colorado Iron Works, Denver, Colo., about 1890; Figs. 174 and 175 represent the standard furnace of the International

[1] Daggett Hofman, *Min. Ind.*, 1905, XIV, 409.

[2] Hahn-Eilers-Raymond, *Tr. A. I. M. E.*, 1871–73, I, 108.
Arents, *Eng. Min. J.*, 1885, XL, 287.

[3] Devereux, 1887, December 6, No. 374239; 1888, April 17, Nos. 381118 and 381119; June 12, No. 384349; 1889, July 23, Nos. 407335, 407336, and 407337; December 17, Nos. 417314 and 417315; 1890, May 6, No. 427058.
Konemann, 1888, October 9, No. 390785.
Wilson, 1889, May 21, No. 403815, and others.

[4] Palmer, *Tr. A. I. M. E.*, 1914, XLIX, 507.
Dwight, *Eng. Min. J.*, 1916, CII, 671.

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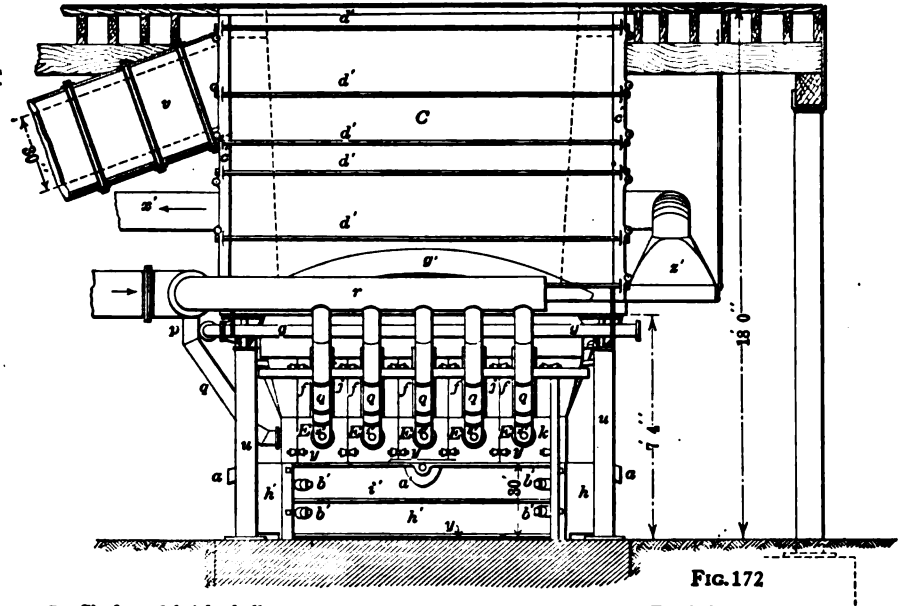

FIG. 172

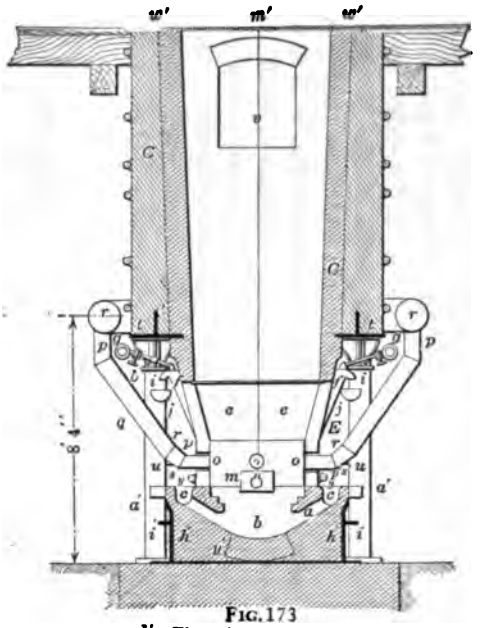

FIG. 173

C	Shaft, red brick shell	*m*	Tap-hole	*d'*	Tie-rods
C'	Shaft, firebrick lining	*n*	Tapping jacket	*g'*	Brick arch
E	Cast-iron side jacket	*o*	Tuyere	*h'*	Crucible castings
a	Slag-spout	*p*	Blast-pipe	*i'*	Strengthening-ribs of crucible-castings
b	Crucible	*q*	Wind-bag	*m'*	Feed-opening
c	Lead-well or basin of Arents automatic tap	*r*	Bustle-pipe	*p'*	Brass nozzle
d	Syphon, inclined channel of Arents automatic tap	*t*	Cast-iron collar, supporting- or carrier-plate	*s'*	Eye or peep-hole
e	Water-jackets	*u*	Cast-iron pillar	*w'*	Top plate
f	Cast-iron water feeder	*v*	Down-comer	*x'*	Pipe leading to fan
g	Lateral water-supply pipe	*y*	Lugs fastened by bolts	*y'*	Bed-plate
h	Water feed-pipes	*a'*	Lead-spout	*z'*	Hood leading into pipe x^1
i	Water overflow-pipes	*c'*	Corner-irons for tie rods		

FIGS. 172 and 173.—Blast-furnace, 36 by 100 in., Colorado Iron Works. Denver, Colo., 1880.

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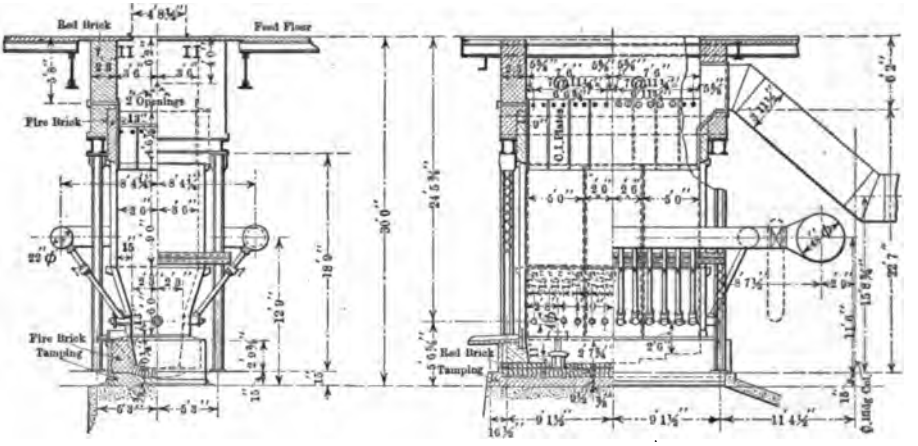
Smelting Co., Tooele, Utah, designed by W. Wraith; Figs. 176–179, one of the furnaces of the Consolidated Mining and Smelting Co. of Canada, Trail, B. C., designed by R. H. Stewart; Figs. 180 and 181 give the details and Figs. 182 and 183 two elevations of the furnace designed by S. James and installed by the Traylor Engineering Co., Allentown, Pa., at the works of the Northport Smelting and Refining Co., Northport, Wash.; Figs. 184 and 185 is the furnace of the Bunker Hill and Sullivan Mining and Concentrating Co. at Kellogg, Idaho, built by the same firm according to the plans of Bradley, Bruff, and Laharthe, San Francisco. These drawings are sufficient to serve as a basis for the discussion of the principal features of modern blast-furnaces. Other illustrations will be given to bring out details not shown with sufficient clearness in Figs. 172–185.



FIGS. 174 and 175.—Blast furnace, International Smelting Co., Tooele, Utah.

In Table 53 are assembled the principal facts of some of the leading blast furnaces in operation at present. Other tables, representing the earlier practice, have been published by Croll,[1] Kochinke,[2] Iles, M. W.,[3] Borchers,[4] Ralli,[5] and Collins, H. F.[6]

103. **Foundation.**—The weight of a lead blast furnace with brick shaft is about 100 tons. The shaft, the heaviest part, is carried by four cast-iron or steel columns. Though the structure as a whole is not very heavy, considerable attention has to be given to the foundation. Its depth will depend upon the character of the subjacent ground. If bed-rock is exposed, this will furnish as good a foundation as can be wished. Next in order comes hard pan, a mix-

[1] *Eng. Min. J.*, 1898, LXV, 639.

[2] *Freiberg. Jahrb.*, 1901, 216; *Min. Ind.*, 1901, X, 430; correction by Cap., *Oest. Zt. Berg. Hüttenw.*, 1902, L, 147; *Min. Ind.*, 1902, XI, 442.

[3] "Lead Smelting," Wiley, New York, 1902, 48, 49.

[4] *Metallurgie*, 1907, IV, 100.

[5] *Rev. Un. Min.*, 1911, XXXIV, 216.

[6] "Lead," Lippincott, Philadelphia, 1910, 312.

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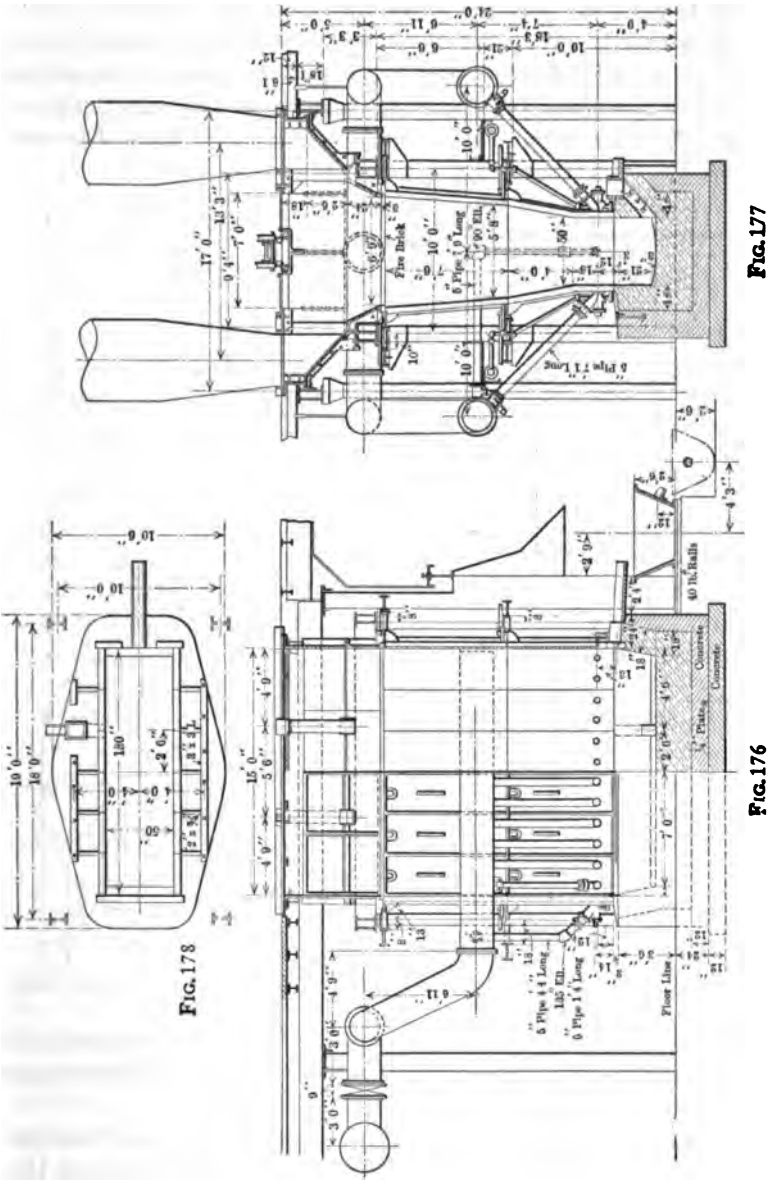

FIG. 178

FIG. 177

FIG. 176

FIGS. 176 to 178.—Blast furnace, Consolidated Mining and Smelting Co. of Canada, Trail, B. C.

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ture of gravel, clay, and sand, which has a bearing power of 4 tons per sq. ft. If there is loose soil or gravel covering bed-rock or hard pan not over 10 ft. deep, it is best to excavate until this is reached; otherwise a depth of 5 ft. will usually be sufficient to start the masonry below frost-line, and to give the foundation the requisite strength. With very loose soil, which has a bearing power of 2 tons per sq. ft., it used to be customary to place on the bottom of the pit two layers of 3- or 4-in. planks spiked crosswise together, and upon that to build the foundation. At present the start is made with a 12-in. layer of cement con-

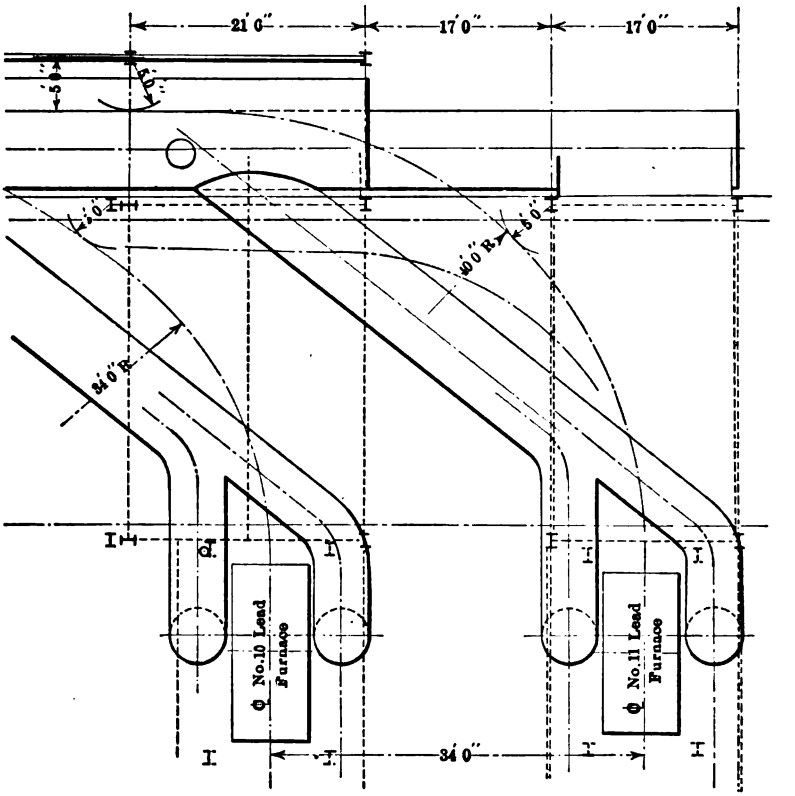

FIG. 179.—Blast furnace, Consolidated Mining and Smelting Co. of Canada, Trail, B. C.

crete which has a bearing power of 4 tons per sq. ft. Whatever method is used, it is essential that the foundation extend from 2 to 3 ft. beyond the bed-plate and the four supporting columns.

The foundation used to be built of undressed rock well rammed into place, the largest pieces being used for corners, and care being taken to fill up the crevices and joints with as many spalls as possible; the whole was well grouted with a mixture of four parts of lime mortar and one part of cement. The top must of course be smooth and horizontal, and was usually made of brick. This method has been replaced in large part by cement concrete, which is preferable and usually cheaper. If one furnace is in operation, and a second one is to be

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erected, the excavated pit is filled with broken slag, and liquid slag is poured in to close the cavities and to cement the lumps. The top is evened and leveled by making shallow rectangular areas surrounded by sand or pieces of structural iron, and filling them with liquid slag. Any ridges or other rough parts are removed by chipping. Filling the excavated space with liquid slag is a simple operation, but not quite as satisfactory as the one outlined, because slag when cooled quickly becomes glassy, is brittle, and cracks readily.

On the foundation is spread a thin layer of clay-mortar upon which is placed the bed-plate (*y'*, Figs. 173, 175, 180, 181), a steel plate about ¼ in. thick, which prevents any lead which has traveled downward through the joints in the brickwork of the crucible, from penetrating into the foundation. Following the example of some copper-matting furnaces, a few lead furnaces have been built with an air-space beneath a cast-iron bottom plate. As there is never an excess of heat in the crucible of a lead furnace, this construction, which increases the loss of heat, has not found favor.

104. Shaft.—The leading points to be considered are the height, the horizontal and vertical sections, the building material, and the disposition of the feeding and gas-withdrawing openings.

105. Height.—This term is intended to designate the vertical distance of the shaft which is thermally and chemically active, *i.e.*, the working height or smelting column. It extends from the center of tuyères to feed-floor, in furnaces from which the gases are withdrawn at the top (Figs. 177, 180, 183); and to the lower edge of the down-take, if the gases are withdrawn beneath the feed. Sometimes there is a difference of 10 ft. between the two, but usually it does not exceed 5 ft. (Figs. 173–175).

About 20 years ago the distance from tuyères to feed-floor used to be from 10 to 12 ft.; it was then increased to 14 and later to 16 ft., which is a common dimension now, although it reaches 20 and even 24 ft. (Sulphide Corporation, Table 51). The increase in height has been necessitated by the greater volume and pressure of blast, required by the enlargement of the distance between tuyères to increase the capacity and the smelting power of a furnace, and by the silicious and calcareous slags which have replaced those running low in SiO_2 and high in FeO. Whereas the blast-pressure used to be from 8 to 10 oz. per sq. in., it ranges at present from 30 to 40 oz., and in the case of the Sulphide Corporation reaches 70 oz. (Table 51).

106. Horizontal Section.—In §102 it was stated that with ore furnaces the oblong rectangular section had superseded the circular. The circular furnace gives excellent results as regards quality of work; there is an even distribution of blast and heat, and the loss of heat by radiation is the least possible, as the furnace offers the largest cross-sectional area for the smallest circumference. The drawback lies in the quantity of the work, which is limited, since the diameter at the tuyère-section ought not to exceed 60 in. With the charge made up largely of fine material, the usual distance was 42 in.; with blast roasted ore this dimension has been raised without trouble to 54 in. although 44 and 48 in. are the more common figures.

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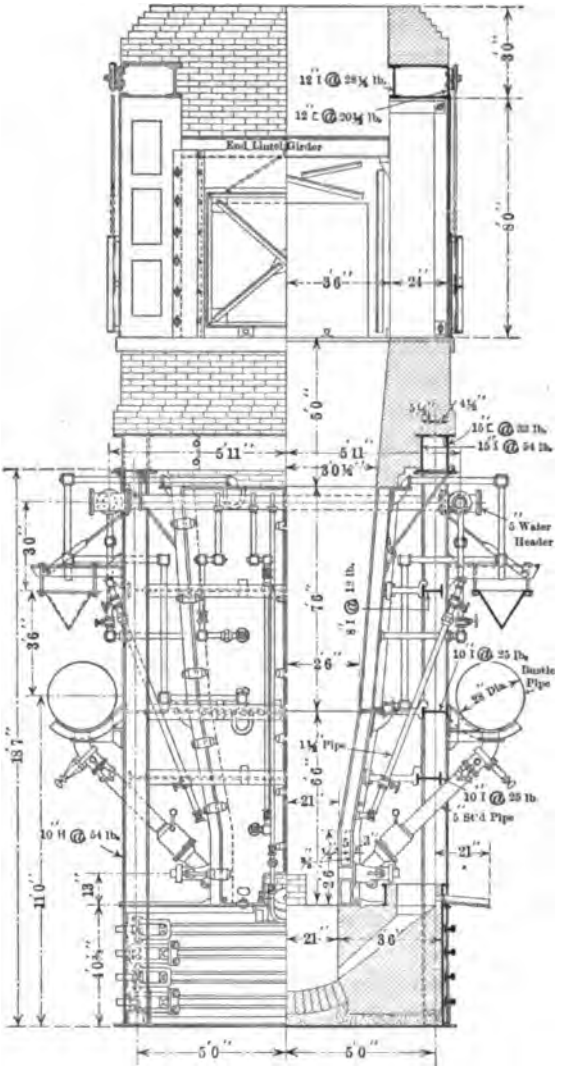

FIG. 180.—Blast furnace, Northport Smelting and Refining Co., Northport, Wash.; details.

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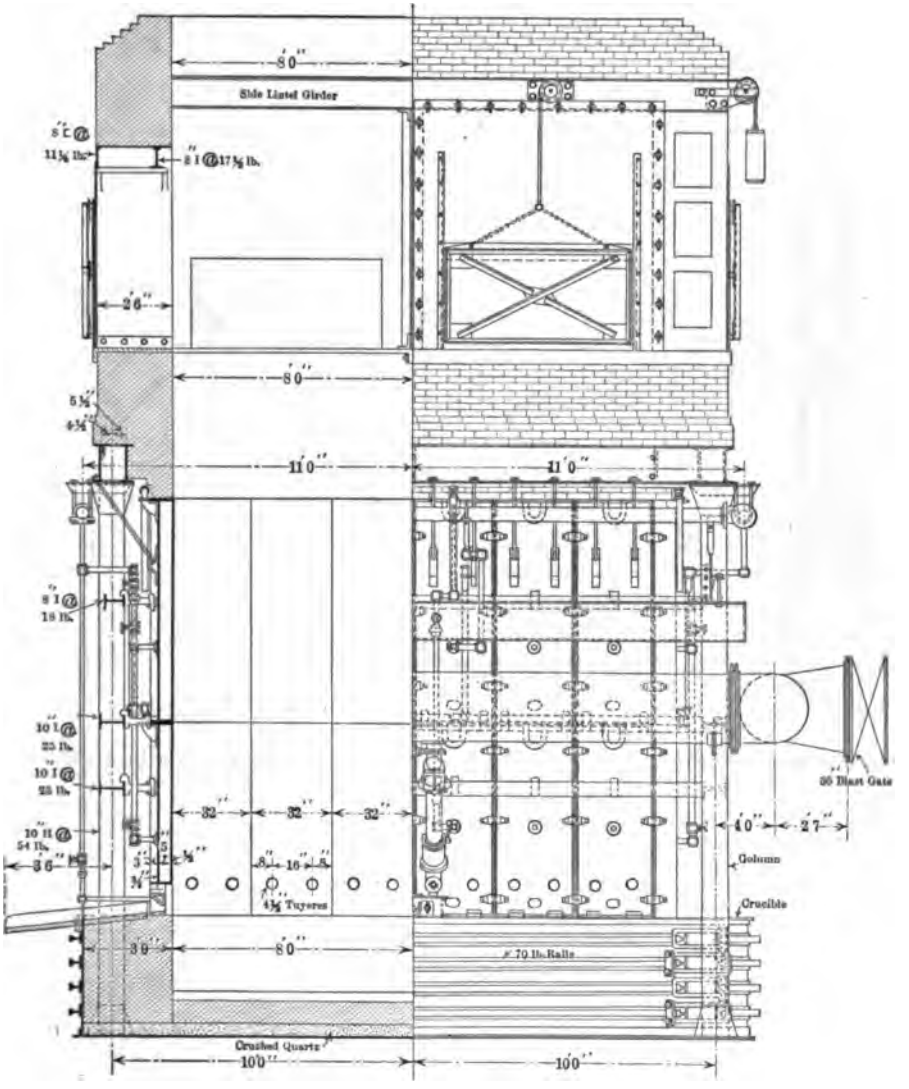

FIG. 181.—Blast furnace, Northport Smelting and Refining Co., Northport, Wash.; details.

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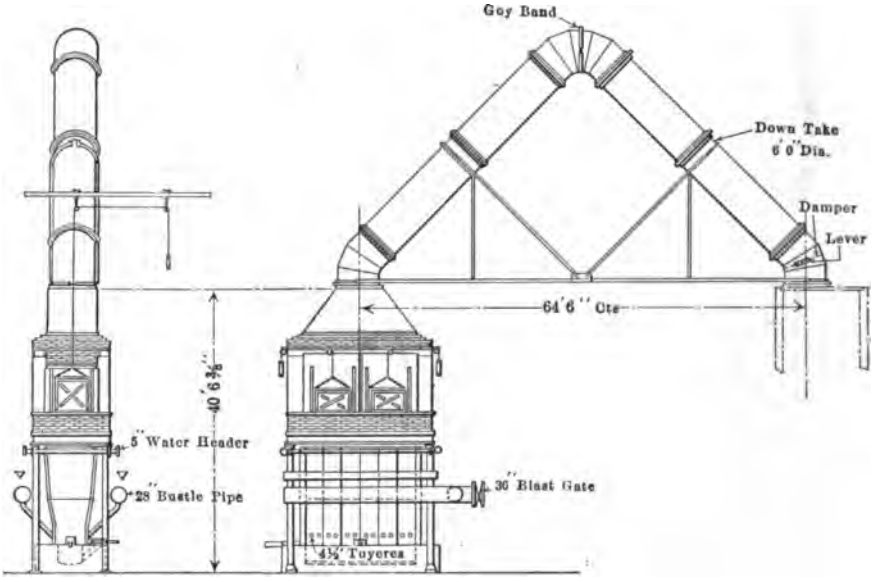

FIGS. 182 and 183.—Blast furnace, Northport Smelting and Refining Co., Northport, Wash.; elevations.

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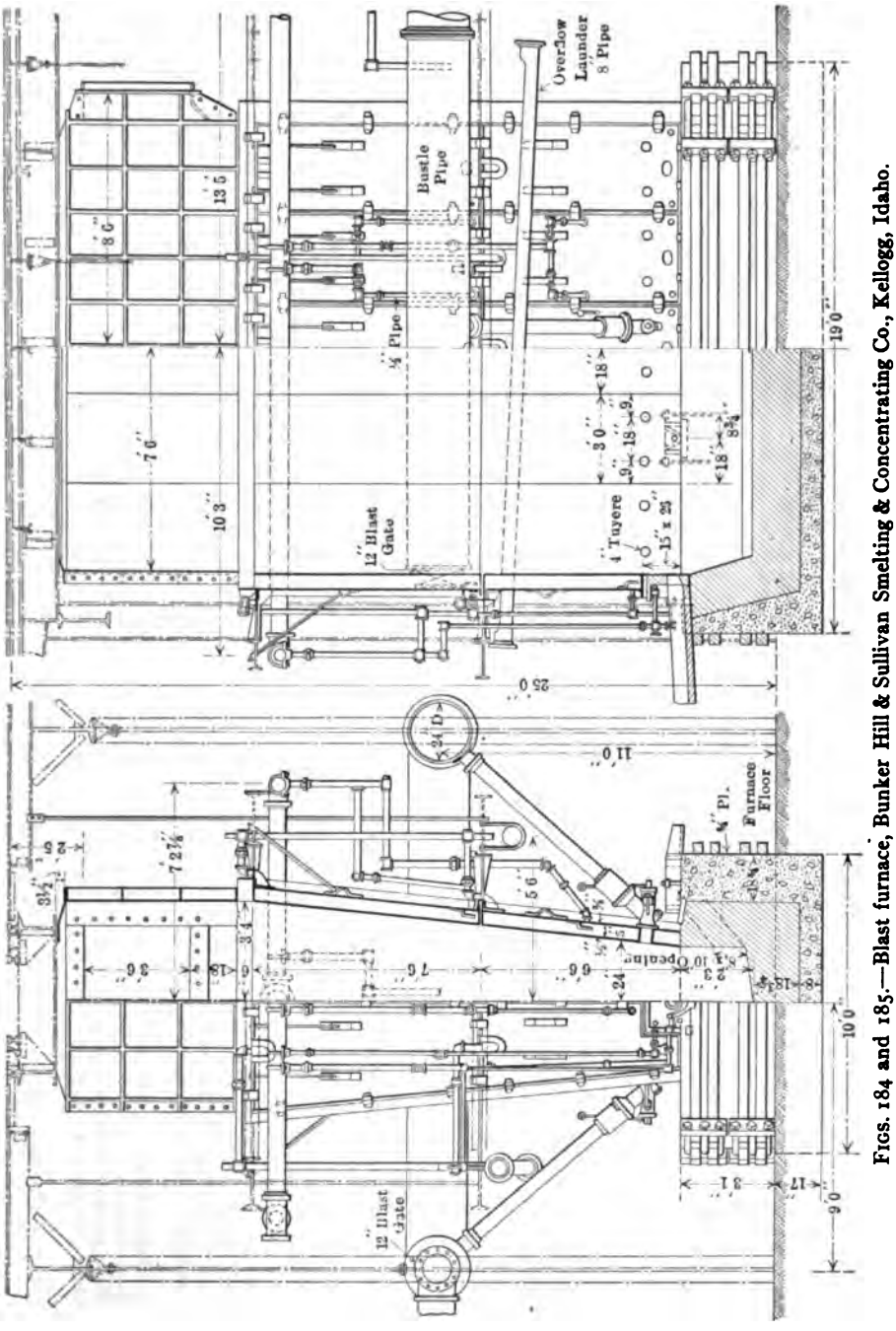

FIGS. 184 and 185.—Blast furnace, Bunker Hill & Sullivan Smelting & Concentrating Co., Kellogg, Idaho.

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TABLE 53.—BLAST FURNACE

	U. S. Smelting Co., Midvale, Utah	Ohio & Colo. S. & R. Co., Salida, Colo.		St. Joseph Lead Co., Herculaneum, Mo.	Int. S. Co., Tooele, Utah	American Smelting & Refining Co.	
Horizontal section at throat (feed-floor), in.	95×160	68×156	88.5×207	84×192	93×180	93×176	84×180
Area at throat, sq. ft.	105.55	73.67	127.22	112.00	116.25	113.66	105
Horizontal section at tuyères, in.	48×160	40×144	48×180	42×192	54×180	48×176	44×144
Area at tuyères, sq. ft.	53.33	43.20	60.00	56.00	67.5	58.66	44
Ratio throat to hearth area	1.98	1.70	2.12	2.00	1.75	1.94	2.4
Height, tuyères to throat (feed-floor), ft., in.	24.8	17.9	17.9	20.0	24.8	24.5.25	21.0
Height, active or working, ft., in.	16.0	17.9	17.9	13.6	13.6	16.0	21.0
Height, tuyères to top of crucible, in.	13	11.5	11.5	18.75	13.00	15.00	16.875
Water-jackets, cast iron or steel	Steel	Cast iron	Cast iron	Steel	Steel	Steel	Cast iron
Water-jackets, number on one side and end	5S, 1E	8S, 3E	11S, 4E	4S, 1E	3S, 1E	6S, 2E	8S, 2E
Water-jackets, height, ft., in.	6.0	4.6	4.6	6.0	6+9=15	5.11	5.1
Bosh, height, ft. in.	4.1	3.6.5	3.6.5	3.11	4.1 (3)	4.1.25	3.2.5
Bosh, in. per ft. height	4.41	.....	.....	3.0	4.0	2.4375	2.1875
Tuyères, No. on a side	10	8	11	12	12	10	9
Tuyères, diameter, in.	4	4 5/16	4 5/16	4	4	4	4
Tuyère-ratio (1)	4.71	5.41	2.67	5.38	4.47	4.28	5.14
Crucible, section at top, in.	48×160	36×140	44×176	42×192	54×180	44×164	44×144
Crucible, section at bottom, in.	30×151	36×140	44×176	42×174	54×180	44×154	36×130
Crucible, depth, in.	32.5	30.0	27.0	25.0	24.0	30.0	27.0
Crucible, holds lb. lead	.....	40,000	60,000	50,000	80,000	80,000	94,000
Lead-well, position, near slag-tapping end	.....	.....	.....	Under 5th tuyère	32" from front	40" from front	33" from front
Lead-well, area of channel, in.	6×6	.....	.....	5×5	8×12	8×10	8×13.25
Lead-well, area of basin, in.	12×12	.....	.....	10×12	8×12–12×16	8.5×12.5	9.25×15.0
Leadwell, height of top of basin above top of crucible, in.	8–12	.....	.....	10	6	5 5/8	6
Forehearth, inside length × width × depth, in.	108×54×12	.....	(2)	67.5×43×25	139.5×60×48	104×50×42	96×49 & 40×40
Forehearth, capacity, cu. ft.	40	99	99	42	232	126.4	98
Waste slag-pots, No.	20	4	6	10	12	7–8	4
Waste slag-pots, capacity each, cu. ft.	28	25	25	25–30	65	27.8	25
Charge (ore and flux), weight, lb.	8,000	2,500	3,750	7,200	7,600	8,000	2,000

(1) Tuyère-ratio = sq. in. tuyère area: sq. ft. hearth area. (2) 96×49.5 top; 91.5×40.5 bottom; depth 40.5. (3) Lower jacket.

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TABLE 53.—BLAST FURNACE—*Continued*

	U. S. Smelting Co., Midvale, Utah	Ohio & Colo. S. & R. Co., Salida, Colo.		St. Joseph Lead Co., Herculaneum, Mo.	Int. S. Co., Tooele, Utah	American Smelting & Refining Co.	
Charge, blast roasted, per cent	30–65	57	57	86	77	37.9	54.8
Charge, in 24 hr., short tons	160–300; av. 240	167	250	240–260	262(4)	220	167
Charge, tons per sq. ft. hearth area, 24 hr	4.52	3.88	4.17	4.25–4.50	3.86	3.97	3.80
Pb, per cent. of charge	11–18	10.0	10.0	38.0	16.54	13.39	7.00
Cu, per cent. of charge	0.7–1.0	1.0	1.0	1.0	0.86	0.63	0.61
S, per cent. of charge	2.5–3.8	4.0	4.0	4.5	3.12	3.8	3.8
Coke, per cent. of charge	11–13, av. 12	14.0	14.0	12.5	11.65(4)	12.82	14.3
Coke, per cent. carbon	85	79.0	79.0	88–90	.....	80.70	76.1
Coke, per cent. ash	11	19.0	19.0	7.0–8.5	13.22	12.90	20.0
Blast, cu. ft. per min	5,200	5,800	8,100	5,000	4,000	6,600	8,000
Blast-pressure, oz	36–42	28–38	28–38	22–32	32–34	36	40
Cooling-water for jackets, gal. per hr	Variable	4,800	6,000	4,000	12,000	18,000	2,700
Men in 8-hr. shift, regular crew	5	6	7	15	66?	3(5)	3
Lead bullion, oz. Ag, Au per ton	80–160	170	170	2.6–3.2	72.9	94.7	157
Speiss, per cent. charge	N. d.	None	None	None	None	None	.....
Speiss, Pb per cent	N. d.	.....	.....	.....	.....	.....	.....
Speiss, Cu per cent	N. d.	.....	.....	.....	.....	.....	.....
Speiss, Ag oz. per ton	Variable	.....	.....	.....	.....	.....	.....
Matte, per cent. of charge	10.0	10.4	10.4	13.8	6.5	9.25	9.5
Matte, Pb per cent	10–18	14.5	14.5	13.0	12.3	16.7	13.4
Matte, Cu per cent	6–12	12.3	12.3	2.5	11.0	11.23	10.7
Matte, Ag oz. per ton	30–50	53.5	53.5	.....	17.7	29.7	39.4
Slag, SiO_2 per cent	34	31.0	31.0	32.3	30.13	32.5	33.4
Slag, Fe(Mn)O per cent	28	34.4	34.4	36.7	39.46	33.24	32.6
Slag, Ca(Mg)O per cent	22	17.9	17.9	18.4	14.17	19.93	22.2
Slag, Al_2O_3 per cent	3	5.6	5.6	3.7	.....	2.3	6.0
Slag, ZnO per cent	5–9	7.8	7.8	5.6	8.36	8.9	3.9
Slag, Pb per cent	0.6–1.2	1.22	1.22	1.4	1.15	1.01	1.06
Slag, Ag oz. per ton	0.5–0.7	S 1.9, Cu 0.20	.....	None	0.34	0.44	0.46
Slag, specific gravity	N. d.	3.63	3.63	3.6	3.81	3.12	.....

(4) Excluding secondaries. (5) One furnace-man and two tappers.

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TABLE 53.—BLAST FURNACE—*Continued*

	American Smelting & Refining Co.	Consol. Min. & S. Co. of Canada, Trail. B. C.	Sulphide Corporation, Cockle Creek Wks., Boolaroo, N. S. W.	Northport S. & R. Co., Northport, Wash.	Bunker Hill Mining & Concentrating Co., Kellogg, Idaho
Horizontal section at throat (feed-floor), inches	84×180	81×180	123×201	84×192	80×180
Area at throat, sq. ft.	105	101.25	171.67	112	100
Horizontal section at tuyères, in.	44×162	50×180	60×138	42×192	48×180
Area at tuyères, sq. ft.	49.5	62.5	57.5	56.0	60
Ratio throat to hearth area	2.1	1.61	2.98	2.0	1.33
Height, tuyères to throat (feed-floor), ft., in.	21, 0	19, 3	29, 8	17, 9	20, 8
Height, active or working, ft., in.	21, 0	13, 3	24, 2	17, 9	16, 0
Height, tuyères to top of crucible, in.	16	12	16	10	14
Water-jackets, cast iron or steel	Steel	Steel	Cast iron	Steel	Steel
Water-jackets, number on one side and end	4S, 1E	2×6, 3S (7)	8S, 3E	2×6S, 4E	2×5S, 2E
Water-jackets, height, ft., in.	5, 0	(6, 6)+(7, 6)=14	6, 6	(6, 6)+(7, 6)=14	(6, 6)+(7, 6)=14
Bosh, height, ft. in.	3, 2	4, 0	4, 4	4, 0	14
Bosh, in. per ft. height	2.2875	2.25	4.2	2.25	1.25
Tuyères, No. on a side	12	12(8)	7	12	10
Tuyères, diameter, in.	4	4	3	4.5	4
Tuyère-ratio (1)	6.09	5.02	1.72	5.38	4.19
Crucible, section at top, in.	44×162	50×180	54×132	42×192	45×180
Crucible, section at bottom, in.	24 & 35×148	50×168	54×132	26×173	45×168
Crucible, depth, in.	15.0 & 27.5 (6)	24.0	20	36	25.5
Crucible, holds lb. lead	50,000	85,000	38,500	90,000	82,400
Lead-well, position, near slag-tapping end	31" from front	60" from front	Center of side	Center of side	54" from front
Lead-well, area of channel, in.	8×13.5	8×8	4×4	5×17	8×10
Lead-well, area of basin, in.	9×14	8×8	4×8	11×43	8×10
Lead-well, height of top of basin above top of crucible, in.	6	4	10	9	6
Forehearth, inside length × width × depth, in.	.....	60 diam.×30	Overflow pot 1 ton capacity	120×52×42	Two, circular, 6' 3" top, 3' 8" bottom, 2' 7" depth
Forehearth, capacity, cu. ft.	98	30(9)	.....	141	50
Waste slag-pots, No.	4	.....	.....	.....	3
Waste slag-pots capacity each, cu. ft.	25	.....	3	70	130
Charge (ore and flux), weight, lb.	.....	5,400	2,250	6,000	3,600–4,000

(1) Tuyère-ratio = sq. in. tuyère area: sq. ft. hearth area. (6) At Lead-well. (7) One lower +2 upper. (8) One at back. (9) Two forehearths changed every 24 hr.

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TABLE 53.—BLAST FURNACE—*Concluded*

	American Smelting & Refining Co.	Consol. Min. & S. Co. of Canada, Trail, B. C.	Sulphide Corporation, Cockle Creek Wks., Boolaroo, N. S. W.	Northport S. & R. Co., Northport, Wash.	Bunker Hill Mining & Concentrating Co., Kellogg, Idaho.
Charge, blast roasted, per cent		70	75	70	60
Charge, in 24 hr., short tons		200	336	230	250–285
Charge-tons per sq. ft. hearth area, 24 hr		3.2	5.81	4.11	4
Pb, per cent. of charge		15–22	33	25	36–40
Cu, per cent. of charge		0.40	None	Small	
S, per cent. of charge		3.5	2	4	
Coke, per cent. of charge		14.0	12	14	13
Coke, per cent. carbon		82.5	83	81	
Coke, per cent. ash		15.0	12	14	15–22
Blast, cu. ft. per min			7,500	7,000	8,000–10,000
Blast-pressure, oz		28	70	30–40	38–42
Cooling-water for jackets, gal. per hr			12,000	43,428	
Men in 8-hr. shift, regular crew		6	22	8	
Lead bullion, oz. Ag, Au per ton		120	80	80	
Speiss, per cent. charge		None	None	None	None
Speiss, Pb per cent					
Speiss, Cu per cent					
Speiss, Ag oz. per ton					
Matte, per cent. of charge		7.0	None	9	
Matte, Pb per cent		18.0		16	12–15
Matte, Cu per cent		8.0		3	2–4
Matte, Ag oz. per ton		35.0		30	25
Slag, SiO_2 per cent		32.0	23	32–35	33
Slag, Fe(Mn)O per cent		29.0	38	34–37	33
Slag, Ca(Mg)O per cent		20.5	17	18–20	16.5
Slag, Al_2O_3 per cent		6.5	5	3	5.7
Slag, ZnO per cent		10.0	12	3	5.0
Slag, Pb per cent		1.5	0.6	0.7	0.8
Slag, Ag oz. per ton		0.75	S3	0.6	0.38
Slag, specific gravity		3.68			

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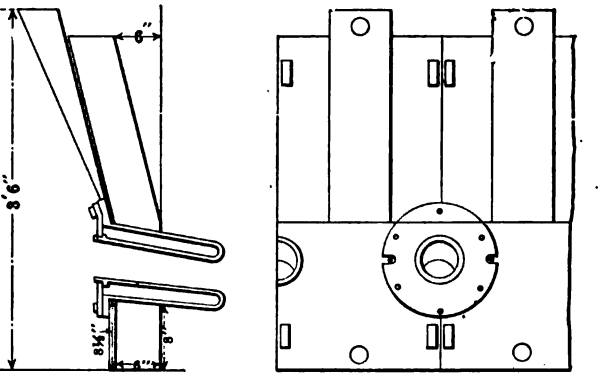
In the early eighties E. P. Mathewson placed water-cooled tuyère-nozzles, Figs. 186 and 187, in the tuyère-openings of the water-jackets so as to project 6 in. into the 60- by 120-in. furnaces of Pueblo, Colo., and thus reduced the distance between tuyères to 48 in. His smelting power with six 4-in. tuyères on a side and a blast-pressure of 2 lb. per sq. in. was 200 tons of medium-open charge, containing Pb 10 and matte-fall 3.5 per cent., with 10 per cent. coke; the slags made being SiO_2 36, FeO 32–33, CaO 18 per cent.; and SiO_2 36, FeO 28–29, CaO 20 per cent.

The reason for limiting the distance between tuyères is the great blast-pressure required to penetrate the charge, which would cause excessive volatilization of lead and reduction of Fe_2O_3. For large quantities of ore the oblong form



FIGS. 186 and 187.—Mathewson water-cooled tuyère-nozzle.

is therefore the proper one, as the tuyère-section can be enlarged by making the furnace longer without increasing the distance between tuyères. Thus the length of oblong furnaces was at first doubled from the old 60 to 120 in.; later it was further increased to 160 and 180 in. (see Table 51). It is an open question whether furnaces of such lengths are not too long. A normal crew can serve a furnace 152 in. long; if this dimension is increased, extra men are necessary, and the additional tonnage obtained is not equivalent to the increase in cost of labor. Many furnace-men hold that 152 in. is the greatest economic length of furnace with a single slag-tap; others draw the line at 192 in. It is, of course, possible to follow the example set by copper-matting furnaces[1] and have larger furnaces with a double complement of slag-taps and lead-wells, but the smelting of lead ores is a more delicate matter than that of copper ores. In case of unsatisfactory work, it is usually cheaper to blow down a furnace and start fresh instead of attempting to nurse it back to normal work. With a copper furnace this is done more quickly than with a lead furnace, and involves less loss of metal.

The larger and correspondingly higher furnaces have shown a greater smelting power than was expected from the increased hearth area, and the increased tonnage has been accompanied by a lower fuel consumption.

[1] See Hofman, "Metallurgy of Copper," McGraw-Hill Book Co., New York, 1918, 178.

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107. Vertical Section.—Most oblong furnaces have boshes on the sides and with them a contracted tuyère-section. This last secures a more perfect and rapid combustion, and thereby a concentrated and intensified heat, which results in a quicker fusion and a more complete decomposition of sulphide and arsenide of lead. If somewhat higher up the width of the furnace is suddenly enlarged by the boshes, the zone of fusion will be narrowed; further the gases generated at the tuyères will be evenly diffused, thus checking the velocity of their upward travel; by giving up their heat they prepare the charge for the subsequent smelting and decrease the amount of flue-dust.

The ends of a furnace are usually vertical although some furnaces have boshes at ends as well as at sides. The general experience has been that end-boshes favor the formation of accretions and thereby the hanging of the charge.

The amount of bosh, or the flare of the jackets, that is, the number of horizontal inches per foot of height, is seen in Table 53 to range from 2 to 4. With charges consisting largely of fine materials an ample bosh was justified, as it helped to support the charge and exposed it for a longer period of time to the heating and reducing effects of the ascending gases. Since the advent of blast roasted ore the flare has been greatly reduced. Thus Palmer[1] states that in a furnace with an amount of bosh of 4.2 in. there was noticed, in blowing in, a hanging of the charge at the boshes causing irregularities in their descent, which disappeared after 1 or 2 weeks, whereupon work became normal. After blowing down, it was found that the furnace lines had been changed to form a gently flaring shaft with a flare of about 1 in. to the foot. The new furnace of the Bunker Hill and Sullivan smeltery, Figs. 184 and 185, embodies this idea. Anderson[2] states that of two furnaces at Midvale, Utah, 48 in. at tuyères, running on the same coarse charge, the one with a bosh of 2.2 in. ran hotter, and gave a smaller amount of matte (owing to the expulsion of 50 per cent. of the S) and a larger yield of lead bullion than the other, with a bosh of over 4.2 in., which oxidized about 33 per cent. of the S-content. He further calls attention to the accepted fact that furnaces with incrusted sides frequently do better work than when the sides are clean, and goes as far as to suggest that the sides be drawn in toward the throat in building a furnace as is the case with the iron blast furnace.

108. Building Material.—Shafts used to be, and still are largely of brickwork; most new-built furnaces are water-jacketed from top of crucibles to a short distance beneath the feed-floor.

The brick shaft is made of common brick, *C*, and lined with fire-brick, *C'*, Fig. 173, up to the feed-floor; sometimes an expansion space is left open between the two. At first it was erected on four supporting plates, *t*, which were fastened to the capitals of the hollow cast-iron pillars, *u*, Fig. 173, but the pillars were thrown out of plumb by the unequal expansion of brickwork and cast-iron plates, and thus the safety of the shaft endangered. Later a set of three I-beams on each side of the furnace formed the support of the cast-iron plates;

[1] *Tr. A. I. M. E.*, 1914, XLIX, 509.
[2] *Op. cit.*, 523.

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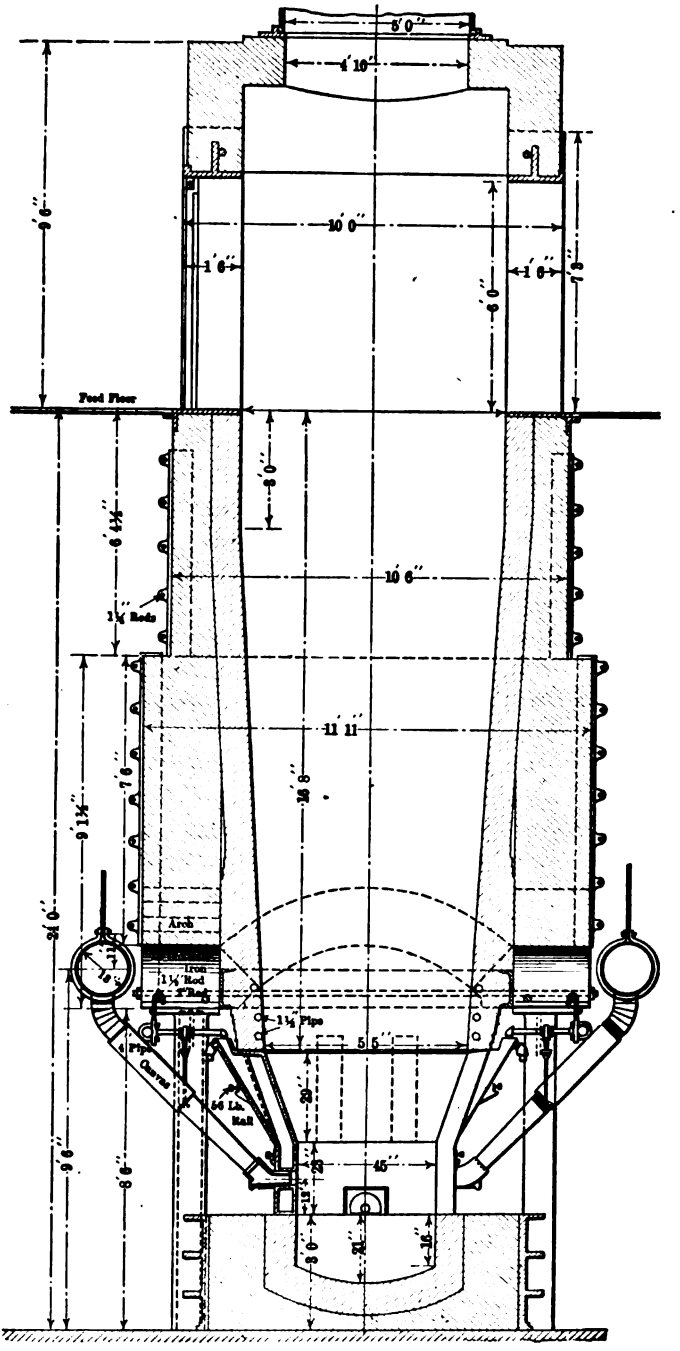

FIG. 188.—Blast furnace, Germania Lead Works, Salt Lake City.

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they were bolted together and screwed to the capitals of the pillars. The cast-iron plates were not fastened to the beams, but rested freely upon them. By this arrangement, supporting plates and shaft could expand independently of one another without endangering the stability of the shaft. The next improvement was the introduction of brick arches which supported the walls of the shaft and threw the weight upon the pillars; the lateral thrust was taken up by cast-iron L-shaped plates firmly bolted together. This arrangement, common at present, is shown in Fig. 188. It gives the shaft-support of the furnace, 45 by 100 in. at tuyères and 84 by 144 in. at throat with a height of 20 ft. 10 in., of the former Germania Lead Works of Salt Lake City, designed by H. V. Croll.[1] Heavy brick arches carrying the shaft are sprung between suitable skew-backs attached to the columns and connected by 1½- and 2-in. tension rods.

The walls of modern brick furnaces are made very thick at the bottom, 32.5–39.0 in., in comparison with those of earlier date, 17 in., and decrease in thickness toward the feed-floor either uniformly or in steps. It is believed that this thickness of wall causes a considerable saving in fuel.

The shaft, Fig. 173, is braced with tie-rods, *d'*, secured in corner-irons, *c'*. In more recent furnaces the shaft is secured by horizontal I-beams which are backed by iron plates and tied by rods and turn-buckles.

Brick shafts have the disadvantage that the brickwork is attacked by the components of the charge which usually adhere to it, form accretions troublesome to remove, and penetrate into it, and weaken it. Thus there are frequently found in the unaltered brickwork veinlets of galena;[2] the bricks are more or less corroded; again, they may have been completely changed by the charges.[3] It is true that a brick wall conserves heat better than a water-jacket. The amount of heat lost through jacketing the upper part of the furnace is large. Thus, Wm. Wraith[4] found that with the blast furnaces at Tooele, Figs. 174 and 175, the upper jackets of a furnace required 159 gal. water per min., when entering at 154.4° and leaving at 165.7° F.; this corresponds to 601.18 liters of water at 68° C. raised 6.3° C. = 3700 Cal. Nevertheless, the advantages which jackets offer: small formation of accretions, ease of their removal as they do not attack the walls, and especially the short time required for blowing out and blowing in again a furnace that is out of order, have apparently more than balanced the disadvantages, and have been the cause of their replacing brickwork. According to Wraith,[5] at Tooele, the average time occupied in barring down and repairing brickwork with a double tier of jackets is 2.57 days, with a single tier 4.59 days; the reduction in tonnage per year due to lost time is 1,659.3 tons with a double, and 2268.0 tons with a single tier of jackets.

An early example of a water-jacket shaft is that of the Zeehan and Dundas

[1] *Eng. Min. J.*, 1898, LXV, 641.
[2] Stulzer, *Zt. prakt. Geol.*, 1908, XVI, 119.
[3] Massia, *Oest. Zt. Berg. Hüttenw.*, 1918, LVIII, 391.
[4] Private notes, July, 1916.
[5] Private communication, October, 1916.

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Works, Tasmania,[1] which had wrought-iron upper and cast-iron lower jackets. In all furnaces there is in use a double tier of jackets, the lower enclosing the smelting zone, the upper the shaft proper. The upper jackets usually do not extend to the feed-floor, Figs. 184 and 185, but terminate from 5 to 10 ft. below to make room for brickwork alone, Figs. 180 and 181; or for brickwork protected by cast-iron plates, Figs. 174 and 175, to take up the impact of the charges that are fed mechanically; or for cast-iron plates alone, Figs. 176 and 177.

The side-jackets are 2.5–3 ft. wide and about 7.5 ft. long, their number depending upon the length of the furnace. In the furnace of Tooele, Fig. 175, there are two side-jackets 5 ft. long, and two 2 ft. 6 in.; they are 21 and 24 in. wide. The ends of a furnace usually have each two vertical jackets. Side- and end-jackets are usually suspended by hangers from I-beams (mantle-frame) which support the superstructure of the furnace and the charging floor. They are joined by lugs and bolts, and are braced by horizontal I-beams which are bolted together at the ends with connection angles. The jacket-plates, usually of flanged steel, are joined by riveting or preferably by welding. The inner plates are of heavier material, ½–⅞ in., than the outer, ⅜ in., to counteract warping or buckling.[2] The outer plates have riveted to them distance-pieces which support the inner; the latter thus offer a smooth and strong surface to the descending charge, and resist corrosion better than the older plates with their stay-bolts. The inner plates are more readily corroded than the outer.[3] The water-space is 4.5–5 in.; the water-inlets are near the bottoms and the outlets near the tops; the inlet-pipes are provided with elbows so that the water may be delivered in a downward direction and not impinge upon the inner plates.

109. Feeding and Gas-withdrawing Openings.[4]—All lead furnaces have an open throat, as the issuing gases have no fuel value. The idea of closing the throat with cup and cone, as with the iron blast furnace, and feeding mechanically was suggested some 30 years ago by Hahn[5] and tried at a later date,[6] but has not been successful, as the heat crept up and gave a hot top.

Open-top furnaces used to be fed almost exclusively by hand, and still are to some extent. With the increase of size of furnaces and the use of blast roasted ore the tonnages of furnaces have so increased that hand-feeding has been largely replaced by mechanical feeding. Whichever method may be used, the charge has to be fed either from the top or the sides. With top-feeding, the gases may be withdrawn from the side or the top; with side-feeding they are invariably withdrawn from the top.

In Fig. 173 the rectangular feed-opening, m', in the top-plate, w', is 20–24 in. wide and 60 in. long, or may extend to the ends of the furnace. The charge

[1] Beardsley, *Tr. A. I. M. E.*, 1882–83, XXI, 575.
Editor, *Eng. Min. J.*, 1893, LVI, 523.
[2] Rice, Straightening Warped Jackets, *Eng. Min. J.*, 1911, XCI, 653.
[3] Lee, *Tr. A. I. M. E.*, 1907, XXXVIII, 877; discussion, p. 878, and 1908, XXXIX, 806.
[4] Dwight, *Tr. A. I. M. E.*, 1902, XXXII, 353.
[5] *Min. Res. U. S.*, 1882, p. 343.
[6] Dwight, *Tr. A. I. M. E.*, 1902, XXXII, 369.

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is fed by hand, and the gases are withdrawn beneath the floor through the down-comer, *v*.

Fig. 189 represents the side elevation of the top of the hand-fed furnace of the former Germania Lead Works,[1] which had a throat area of 7 by 12 ft. = 84 sq. ft. On either side of the furnace is an opening 12 ft. long by 8 ft. high through which the charges are introduced; the opening is divided in two by a vertical 50-lb. rail to which are attached guides. On the feed-floor is a sill, 6

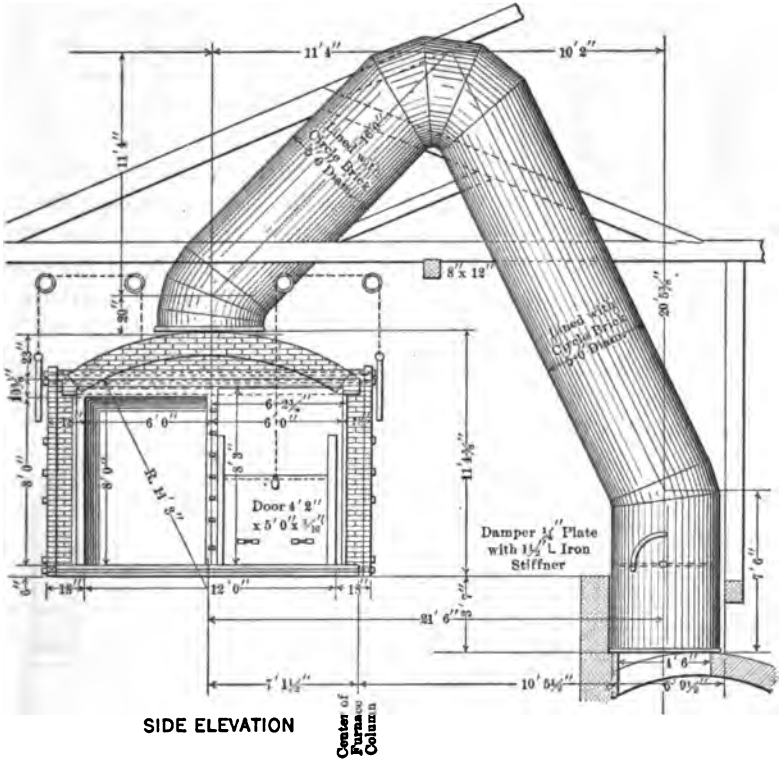

FIG. 189.—Top of hand-fed blast furnace, Germania Lead Works, Salt Lake City, Utah.

in. high, which forces the feeder to lift his charge by the shovelful and prevents his shoving it in; it thus favors the regular prescribed methods of distribution of charge. The gases are withdrawn from the top through a goose-neck which terminates in the roof of the dust-flue.

Terhune patented[2] a device to secure distribution of coarse material near the center and fine near the walls of the furnace. It consists of two grizzlies, one to a side, sloping downward from the feed-doors toward the center, which have tapering bars and form slots narrow near the walls and wide toward the center. A charge dumped onto a grizzly will first drop the fine near the walls

[1] Croll, *Eng. Min. J.*, 1898, LXV, 641.

[2] U. S. Patent No. 585297, June 29, 1897.

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and later the coarse toward the center as it glides down the grizzly. A similar device has been patented by Dwight.[1]

Practical experience has shown that furnaces fed by hand from the top do not work as well as when fed from the sides. Dwight[2] calls attention to the fact that on a shovel, containing fine and coarse ore, the fine parts predominate on the bottom and the center, and the coarse on the top and sides. If the contents are thrown from the shovel, the coarse travels faster than the fine. In feeding a furnace from the top, Fig. 190, the coarse will prevail along the sides, and the fine along the center; in feeding from the side, Fig. 191, the reverse is the case. In a blast furnace the ascending gases meet less resistance near the furnace walls than near the center, and therefore have a tendency to rush up at the sides. In doing this, they overheat the charge near the sides and do not exert the necessary thermal and chemical effect upon the central portion of the charge. Further, the high velocity and temperature of the gases at the sides causes an excessive amount of dust and fume to be formed. Hence, side-feeding by hand is preferable to top-feeding.

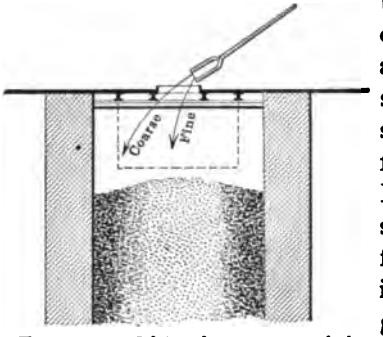

FIG. 190.—Blast furnace, top-fed, by hand.

As to the withdrawal of gases, there cannot be an even ascent through the charge, if the downcomer with its suction is placed at one end of the furnace as in Figs. 172 and 173; in addition to this, the gases, being drawn off from the top of the charge, carry off all the dust they have entrained.

In a hooded furnace, Fig. 191, the gases pass off more evenly from the top of the charge, and their velocity is retarded by air entering the feed-doors; this causes them to drop dust, which falls back onto the charge. Side-feeding is therefore preferable to top-feeding as regards the ascent of the gas-current. Objection is made that it requires a stronger chimney-draft, but this is not necessarily the case, as in regular work the sliding doors of the feed-opening are kept close to the sill, leaving just enough space to permit an even distribution of the charge. Only

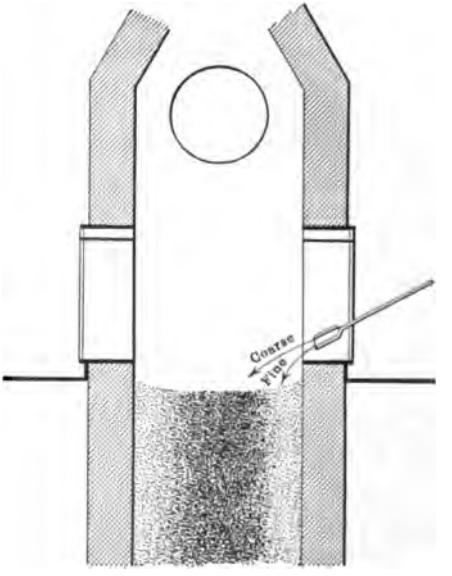

FIG. 191.—Blast furnace, side-fed, by hand.

[1] U. S. Patent No. 959484, May 31, 1910.
[2] *Tr. A. I. M. E.*, 1902, XXXII, 367.

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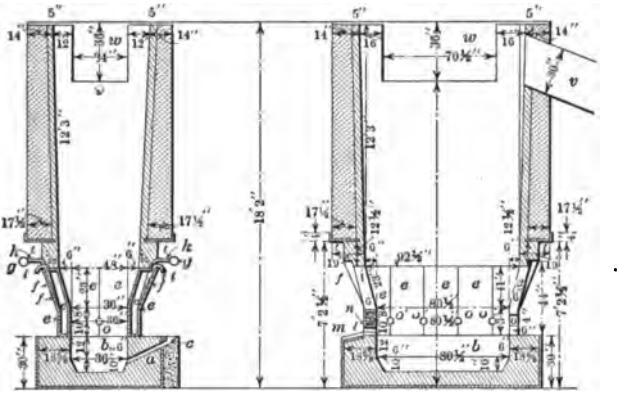
in barring down wall-accretions are the doors raised fully. There is left then an open space, 8 ft. high, which greatly favors the work. With a top-feed the top-plates have to be removed before work of this character can be carried out with any degree of success.

Another device for top-feeding of charge is the Pfort curtain or thimble, which was used first with iron blast furnaces in 1842;[1] with lead furnaces at La Pise, France, before 1868,[2] and at Freiberg, Saxony, about 1869.[3] In 1883



FIGS. 192 and 193.—Blast furnace, Omaha and Grant Smelting and Refining Co., Denver, Colo., 1883.

the furnaces of the Omaha and Grant Works, Denver, Colo.,[4] Fig. 192-193, were provided with this curtain. It consists of an iron shell, *w*, suspended from the top-plates in such a way as to leave room between it and the walls for the gases, which pass off through the flue, *v*, into the dust chamber. The charge fills the inside of the curtain up to the feed-floor. This arrangement is very satisfactory as regards the introduction of the charges and the absence of smoke on the feed-floor. Its drawbacks, however, are so great that it has been discarded. The charge, Fig. 194, in its descent to the downcomer separates more or less into coarse and fine; the coarse rolls toward the wall while the fine remains at the center. This automatic sorting is similar to that taking place in the quartering-method of sampling when, for the purpose of mixing, the ore is coned; the coarse collects toward the rim of the cone, and the fine remains in the center. Two other disadvantages are, that the manner in which the charge sinks can be observed only by allowing it to drop below the thimble; and that for barring down wall-accretions the thimble has to

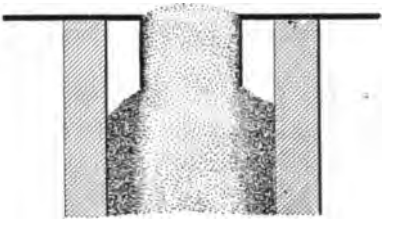


FIG. 194.—Pfort curtain.

[1] Percy, J., "Iron and Steel," Murray, London, 1864, p. 469.
[2] Grüner, *Ann. Min.*, 1868, XIII, 373.
[3] Piltz furnace, *Berg. Hüttenm. Z.*, 1869, XXIX, plate IX.
[4] Frost, *Eng. Min. J.*, 1883, XXXV, 163.

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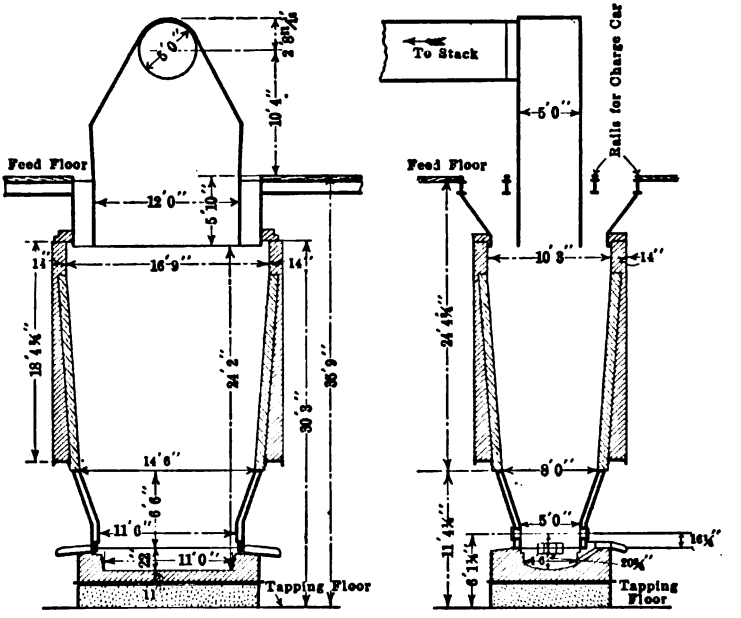
be removed. The withdrawal of gases from the end of the furnace has been taken up above. The areas of ore- and gas-spaces are given in Table 54.

Another method of top-feeding is by means of the Darby tube, a pipe suspended in the center of the furnace, for the withdrawal of the gases, around which the charge is fed. This device, used with the iron blast furnace about 1858,[1] was applied in the Harz Mountains in circular lead blast furnaces about 20 years later[2] and is in operation with several oblong Continental European furnaces, *e.g.*, at Hoboken-les-Anvers, Belgium;[3] Monteponi,[4] Sardinia; in Australia with the Port Pirie furnaces;[5] and in the United States at the Selby Lead Works, Selby, Cal.



FIGS. 195 and 196.—Blast furnace, Sulphide Corporation, Boolaroo, N. S. W.

The furnaces of the Sulphide Corporation, Boolaroo, N. S. W., and the Selby Smelting and Lead Co., Selby, Cal., may serve as examples.

The furnace of the Sulphide Corporation, Figs. 195 and 196, with a throat 16 ft. 9 in. by 10 ft. 3 in. = 171.68 sq. ft., has an oblong thimble, 12 by 5 ft. = 60 sq. ft., for the withdrawal of gases, which extends 5 ft. 10 in. into the furnace and leaves a feeding space 2 ft. 1½ in. wide at the sides and the ends. The top of the furnace is enlarged to make room for tracks on which arrive bottom-discharge feed-cars.

[1] *Berg. Hüttenm. Z.*, 1858, XVII, 263; 1862, XXI, 379.
[2] *Oest. Zt. Berg. Hüttenw.*, 1876, XXIV, 320; *Berg. Hüttenm. Z.*, 1876, XXXV, 336.
[3] *Ann. Min. Belg.*, 1901, VI, 262; 1913, XVIII, 500.
[4] *Oest. Zt. Berg. Hüttenw.*, 1905, LIII, 455; *Eng. Min. J.*, 1905, LXXX, 781.
[5] *Tr. Austral. Inst. Min. Eng.*, 1907, XII, 1; *Eng. Min. J.*, 1907, LXXXIII, 517.

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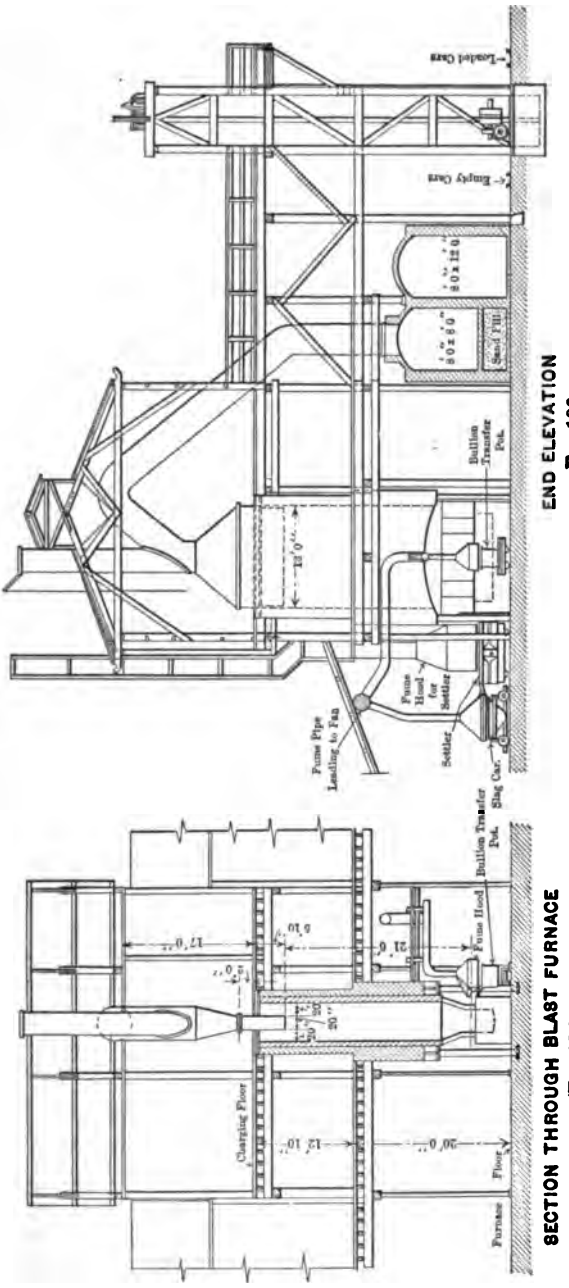
The furnace of the Selby Lead Works,[1] shown in Figs. 197 and 198, has a throat 13 by 5 ft. = 65 sq. ft., an oblong thimble 13 ft. by 1 ft. 8 in. = 21.66 sq. ft.



END ELEVATION
FIG. 198

SECTION THROUGH BLAST FURNACE
FIG. 197

FIGS. 197 and 198.—Blast furnace, Selby Smelting and Lead Co., Selby, Cal.

which reaches to the ends of the furnace and extends downward 5 ft. 10 in.; it

[1] Their work, see Rickard, *Min. Sc. Press*, 1916, CXII, 505.

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leaves a feed-space along the sides of 13 ft. by 1 ft. 8 in. = 21.66 sq. ft. The furnace is hand-fed; fuel and charges are raised from the furnace to the feed-floor by means of a vertical elevator driven electrically. Other details are taken up later.

In experimenting with this form of feed at Murray, Utah, it was found that the charge between thimble and throat sometimes became hung up. At Port Pirie the throat of the furnace, 6 ft. 10 in. wide, has a thimble 4 ft. wide on the feed-floor which extends 3 ft. into the furnace and is contracted at the bottom to a width of 2 ft. 6 in.; it thus corrects all possible danger of the charge becoming wedged fast between side-wall and thimble.

The advantages of this device are that it is suited for hand and mechanical feeding, that the center of the charge is loosened, that the gases are withdrawn along the center-line of the furnace, that the sinking of the charge can be followed, and that shaking up of the charge by means of the steel bar to correct hanging is readily accomplished, as well as the barring down of wall-accretions. This device therefore promises well.

TABLE 54.—PORT AREAS OF FURNACES WITH PFORT CURTAIN AND DARBY TUBE

Item	La Pise(1), circular	Freiberg(2) circular	Omaha, and Grant (3), oblong	Clausthal(4), circular	Sulphide Corporation(5), oblong	Selby Lead Works, oblong	Port Pirie(6), oblong	Hoboken-les-Anvers(7), circular
Area throat, sq. ft.	15.2	46.6	43.5	23.7	171.7	65.0	130.0	18.8
Area gas-opening, sq. ft.	5.0	15.4	16.9	2.1	60*b*	21.6*a*	32*b*	4.9
Area feed-opening, sq. ft.	10.2	31.2	26.6	21.6	111.7	43.4	98.0	13.9
Depth of tube, ft.	1.96	3.08	3.00	2.62	5.83	5.83	3.00	4.26
Ratio gas-opening: feed-opening	2.00	2.02	1.57	10.3	1.85	2.01	3.06	2.83

(*a*) Does extend to ends of furnace. (*b*) Does not extend to ends of furnace.
References: (1) *Ann. Min.*, 1868, XIII, 373. (2) *Berg. Hüttenm. Z.*, 1869, XXVIII 271, plate IX. (3) *Eng. Min. J.*, 1883, XXXV, 163. (4) Kerl, Grundriss der Metallhüttenkunde, Felix, Leipsic, 1881, 39. (5) Private notes. (6) *Tr. Austral. Inst. Min. Eng.*, 1907, XII, 1. (7) *Ann. Min. Belg.*, 1901, VI, 262.

110. Hearth with Arents Automatic or Siphon Tap.—The hearth, or the crucible with its enclosing brick- and iron-work, extends from the foundation to the bottom of the jackets. It is erected on the bed-plate, *y'* in Fig. 172. This sometimes has an angle-iron rim which may enclose the top of the foundation or the bottom course of brick; it may even enclose the iron casing which strengthens the side-walls. Usually the bed-plate is a straight piece of soft steel laid level in cement upon the foundation.

On the bed-plate is placed the iron casing which is to enclose and strengthen the walls of the crucible, as these have to resist the pressure of the lead, charge, and jackets, and the thrust due to expansion. Formerly the casing consisted of cast-iron plates (crucible-castings, *h'*, Figs. 172 and 173) strengthened by ribs, *i*, and fastened together at the beveled corners by tie-rods, *b'*. These castings, even when thick, often developed cracks caused by an uneven distribution of pressure. They were clumsy with small furnaces, and became impossible when these were made larger.

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At present therefore the casing is made of heavy boiler-plate strengthened by I-beams or rails, Figs. 180 and 181, which are tied at the corners by bolts or connecting pieces. In several instances, as at Trail (Fig. 178), Midvale, Pittsburgh, etc., the outer hearth-wall has been made oval and enclosed with boiler-plate ⅜ in. thick; this is sufficiently strong and does away with the cumbersome I-beams or rails.

The crucible is internal, that is, it forms the continuation of the shaft and does not extend beyond it. Slag and matte are tapped at the top of the crucible through tap-hole, *m*, Fig. 173, in tapping-jacket, *n*, at the end of the furnace, and run out over cast-iron slag-spout, *a*. Lead is recovered at the side through cast-iron spout, *a'*, placed a little higher than the slag-spout (see below).

Figs. 172 and 173 show two slag- and two lead-spouts. This arrangement has become obsolete. Tapping from both ends has been carried on to correct the formation of a crust at the back of a furnace with a single tap, where it usually begins and grows toward the front. Two taps were feasible with small furnaces.[1] With the large units common at present, it is preferable for the handling both of charge and of products that one end of the furnace face the dump, and this renders double tapping impossible. There are other ways for counteracting the formation of back crusts. Having two lead-wells is a waste of heat; when one becomes clogged, the other will soon follow suit.

A crucible partly internal and partly external has been given up. The Mathewson[2] fore-hearth was in operation for many years at Pueblo, Colo.

The hearth walls and bottom of crucible are of the best grade of fire-brick. Formerly they rested on the bed plate, Fig. 173. At present a layer of ground brick and clay in the proportion of 3 : 2 (Figs. 174 and 175), or crushed quartz (Figs. 180 and 181) is beaten down firmly to the form of an inverted arch, on which the bricks are placed; this layer allows for expansion of brickwork. In building the side-walls, it is better not to place the bricks in direct contact with the casing, especially if this is cast iron, but to leave open a small space of about 2½ in., and fill this by tamping with brasque, equal volumes of crushed coke and clay. When the brickwork expands upon heating, it will pack the brasque more tightly and relieve the casing from part of the strain. In the two top courses of the side- and end-walls the fire-brick are frequently replaced by magnesite brick, as this material resists the corrosive action of matte much better than fire-brick. In case magnesite brick are not available, fire-brick, *h*, containing much Al_2O_3 are used. Thus two kinds of Laclede brick from St. Louis, Mo., are frequently found, the so-called 70-per cent. with Al_2O_3 66.8, Fe_2O_3 6, SiO_2 30, and the 60-per cent. with Al_2O_3, 60.7, Fe_2O_3 6.9, SiO_2 38. At Trail, B.C., Figs. 176 and 177, a 12-in. layer of concrete encloses the fire-brick walls.

In order to prevent any lead from leaking out at the front through the brickwork, and being carried off with the slag, a water-block with trough-shaped slag-gutter is often built into the brickwork. If any labor has to be performed

[1] Henrich, *Tr. A. I. M. E.*, 1895, xxv, 96.

[2] See preceding edition, page 237.

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in the crucible, a free working opening is easily made by removing the tapping-jacket and water-block.

The depth of the crucible, Table 51, shows a range of from 20 to 30 in.; it is usually nearer the latter than the former figure. The length and width at the top are the same as the distances between the jackets. In order to reduce the amount of lead necessary to fill the crucible, which amounts to 20–40 tons, the brick are frequently stepped down from the sides and ends to the bottom, giving the crucible the form of a truncated pyramid. A decided reduction of the cross-sectional area from front to back is shown in Fig. 175.

The Arents tap,[1] which forms part of the side-wall, consists of an inclined channel, *d*, Fig. 173, from 4 by 4 to 8 by 12 in. in cross-section, which runs from the bottom of the crucible side-wall inside to the top on the outside, and is here enlarged into a dish-shaped basin, the lead-well, *c*, Fig. 173. The tap is placed in the middle of one of the sides, or preferably nearer the front, as it is then more accessible. While the furnace is running, the crucible remains full of lead, or nearly so; the lead in the well stands higher on account of the weight of the charge and the pressure of the blast. In the early furnaces, Fig. 173, run with low blast-pressures, the top of the basin was on a level with that of the crucible. With modern furnaces having blast-pressures of from 30 to 40 oz., the basin has been placed upon the crucible-wall (Figs. 174, 175, 180, 183) in order to raise the level of discharge, and thereby to prevent the lead in the crucible from sinking too far below the level of the tuyères, as this favors the cooling of speiss, matte, and mushy material, and thereby the formation of crusts. The discharge of the basin lies from 4 to 10 in. above the top of the crucible. The height required of the basin to keep the lead in the crucible at the top of the hearth can be calculated,[2] but actual experience will modify the result; in fact, most furnace-men close up the discharge periodically to force the lead in the crucible to rise until some runs out with the slag, and thus insure a complete outflow of speiss, matte, and mush, when the furnace is tapped dry. With charges running high in lead, say 30 per cent., the lead is allowed to overflow into a suitable receiver; with the usual 13 per cent. in silver-lead smelting, it is tapped periodically.

The advantages of the siphon tap are so great that it has been adopted everywhere unless conditions forbid its use. With the older furnaces lead and matte were tapped together from the bottom of the crucible into a shallow basin, and the tap-hole was plugged as soon as slag began to appear. A T-iron with eye was placed upside-down in the molten mass; the matte soon solidified and was raised by means of a chain or wire rope; the adhering lead was allowed to drip off, the cake of matte moved to one side and the lead ladled into molds. Into the crucible of the furnace, freed from lead and matte but still retaining some coke, there dropped an equivalent amount of half-melted charge which had to be raised gradually by fresh lead and matte. When this was the case, tapping began again. These half-melted masses had a chance to adhere

[1] Hahn-Eilers-Raymond, *Tr. A. I. M. E.*, 1871–72, I, 108.

[2] von Schlippenbach, *Metall. u. Erz.*, 1914, XI, 652.

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to the bottom of the crucible and were very apt to be the beginning of bottom-crusts. If these have once started, they are nearly sure to gradually freeze up the furnace. With the Arents tap the crucible is always filled with lead, so that crusts do not form on the bottom of the furnace where they are inaccessible; all crusting takes place on top of the lead, and crusts can be either smelted out or removed by mechanical means.

There are two cases in which the Arents tap can not be used, and tapping from the bottom is necessary. If the charge contains too little lead to replace that contained in the crucible with sufficient frequency, the lead in the crucible will solidify. Charges with as little as 8 per cent. Pb have been run satisfactorily, but they usually contain not less than 13 per cent. With charges running low in lead, bars of lead may be added to the charge. This lead entering the crucible at a temperature of 1000–1200° C. replaces part of the lead in the crucible and heats what remains. But this feeding of metallic lead causes considerable loss in metal, and is practised only in an emergency.

The other case is that of smelting coppery lead ores. Here an alloy of Pb and Cu forms which separates from the lead, adheres to the bottom of the crucible, and fills it as well as the inclined channel of the Arents tap. The trouble is remedied by adding sufficient S in some form or other to make a matte which takes up the Cu. As soon, however, as the matte assays 12 per cent. Cu, the same troublesome separation of Pb–Cu is noticed. In concentrating lead-matte with 12 per cent. Cu or more, the siphon-tap is out of place, and lead and matte have to be removed from the bottom.

111. **Water-jackets.**[1]—These are water-cooled iron shells that enclose the smelting-zone of the furnace to protect it from corrosion by the slag. As long as this zone was enclosed by fire-brick or other refractory material, the smelting had to be so conducted as to prevent the sides from being slagged; with the advent of jackets, this became unnecessary and smelting advanced as regards quality and quantity of work. It is true that a considerable amount of the heat developed in the furnace, perhaps 10 per cent., is carried away by the cooling-water of the jackets;[2] their advantages, however, are so great that in non-ferrous blast furnaces they have replaced everywhere fire-brick and other refractory materials.

In 1885–86 there arose a discussion as to the inventor[3] and the date of the invention of the water-jacket. The earliest mention of water-jackets is made by Overman,[4] who describes and illustrates a refinery furnace the sides of which consisted of water-cooled cast-iron shells, through which water-cooled tuyère

[1] Lang, *Eng. Min. J.*, 1895, LXIII, 89, 137, 188, 258.
Van Zwalmvenburg, *op. cit.*, 352.

[2] Lang, *op. cit.*
Austin, *Eng. Min. J.*, 1897, LXIII, 634; *Min. Sc. Press*, 1908, XCVII, 525. Thermal Balance, §156.

[3] *Eng. Min. J.*, 1885, XL, 56 (Harnickel, Rolker); 75, 109, 318, 335 (Courtis); 109, 142 (Kleinschmidt); 123 (Editor); 180, 214 (Williams); 255 (Hahn); 287 (Arents); 305 (Douglas); 318 (Daggett); 1886, XLI, 2 (Tew).

[4] "Treatise of Metallurgy," Appleton, New York, 1852, 556.

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nozzles protruded into the furnace. Douglas[1] says that J. Williams built near Drontheim, Norway, in 1852 "sectional water-jacket furnaces consisting of a circle of long, narrow water-backs, perforated by tuyère holes." About the year 1865 the same J. Williams erected a number of water-jacket blast-furnaces at Houghton, Lake Superior. According to Arents,[2] Haskell built in 1865 a water-jacket furnace in California. Kerl,[3] in describing the improvements made in smelting in the Harz Mountains, records the introduction in 1864 of water-blocks to cool the hearth and to serve as a support for the water-cooled tuyère nozzles, but these had been used in refinery furnaces for a very long time,[4] and are not to be confounded with water-jackets. The latter were not in use in the Harz Mountains.

Spray-jackets were used at La Pise, France, as early as 1862.[5]

From the statement of Daggett[6] it appears that he erected the first cast-iron jacket at the Winnamuck lead smeltery in 1872.

Cast-iron Jackets.—The original crude cast-iron jackets were improved here and there until they assumed the present form given them by Eilers and Steitz.[7] In recent years soft-steel jackets have begun to take the place of the cast iron; and apparently will replace them entirely in spite of the fact that they cost more and that when worn they form scrap, whereas old cast iron has metal value especially in the neighborhood of a foundry. Usually old jackets are broken up and used as precipitating iron in the charge. Iles[8] noted that corroded cast-iron jackets as well as matte-pots showed values in Ag; thus a corroded matte-pot freed from all adhering material assayed 35 oz. Ag per ton, and cast-iron jackets gave similar values. Steel jackets last longer than do cast-iron, are easily repaired by a blacksmith,[9] are lighter and hence readily removed from the furnace and put again in place. Croll[10] gives a life-record of 14 months for cast-iron jackets of the Globe Works of Denver, Colo., but this is very exceptional; a life of 3 months is nearer the average. Cast-iron jackets are less quickly corroded by salt water than are those of soft steel. Cracks in a cast-iron jacket can be repaired at the works by brazing with a mixture of copper oxide and borax[11] after holes have been bored at the ends of a crack to prevent its extending,[12] or by cutting a dove-tail and caulking with copper; but the result is not always successful.

Furnace men who have exchanged cast-iron jackets while a furnace is running remember the weight.

[1] "Mineral Resources of the United States," 1882, p. 268.
[2] *Berg. Hüttenm. Z.*, 1866, XXV, 316.
[3] *Op. cit.*, 1867, XXVI, 6 and 47.
[4] Percy, "Metallurgy of Iron and Steel," Murray, London, 1864, pp. 584 and 625.
[5] Grüner, *Ann. Min.*, 1862, XIII, 364.
[6] *Eng. Min. J.*, 1885, XL, 316; *Min. Ind.*, 1905, XIV, 409, drawing.
[7] *Tr. A. I. M. E.*, 1915, LII, 735.
[8] Private notes, 1896.
[9] Bromide, *Eng. Min. J.*, 1907, LXXXIII, 344.
[10] *Op. cit.*, 1898, LXV, 639.
[11] Pich process, *op. cit.*, 1906, LXXXI, 228.
[12] Anon, *op. cit.*, 1890, L, 575.

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Cast-iron jackets, *f*, Figs. 172 and 173, are from 3 to 5 ft. high, have walls ⅜–¾ in. thick, and a water-space of 4½–6 in.; the side-jackets, *f*, have a bosh, the end-jackets, *e*, do not. The side-jackets are vertical for a distance of about 18 in. and then spread outward to form a bosh of about 2 ft. in height with a flare of 2–4 in. per ft. A furnace has usually three kinds of jackets, two, *e*, at the ends each half the width of the furnace, and one, *f*, about 20 in. wide, at the sides. Each jacket has its own water-feeder, which begins 8 or 10 in. above the center of the tuyère and extends 3 or 4 in. above the top of the jacket, where it is about 4 in. wide. This insures complete filling of the jacket. The feeder was always cast in one piece with the jacket; later it was frequently a separate casting, fastened with screws or bolts and made water-tight with a gasket. Cooling-water is admitted through the water-feed pipe, *h*, to which is attached a hose to conduct downward the cold water so that the larger part may pass by the rising warm water which is discharged through overflow pipe, *i*, delivering into water-trough, *j*, which empties into a waste-pipe. Jackets have hand-holes to remove scale and mud; they are joined top and bottom by bolts, *y*, passing through lugs cast in the jackets. The tuyère-opening, 3–4 in. in diameter, is in the center of the jacket, 10–12 in. above the bottom. The open space between the tops of jackets and shaft-walls is filled with a course of fire-brick. As this material is frequently slagged wherever smelting takes place above the jackets, two rows of water-cooled pipes are imbedded in the brick-work; in some instances water-cooled iron boxes have been suspended from the capitals of the iron columns.

Beside the regular jackets there is used a tapping-jacket, *n*, Fig. 173. This is a water-cooled casting, *e.g.*, 26 by 14 in. and 3.5 in. deep, made of cast iron or cast steel, which has 6.5 in. above its lower edge a conical tap-hole 2.5 in. in diameter and 1 in. deep. The lower edge of the jackets is placed 4 in. beneath the upper edge of the crucible-casing to prevent at the front all leakage of lead from the crucible, which is otherwise difficult to stop. A cast-iron tapping-jacket may last only 7–8 days with large matte-fall, whereas a jacket made of a water-cooled coil of iron pipe imbedded in copper lasts 70–80 days.

Tap-holes used to be made 1.5 in. in diameter; in Fig. 173 the size is 2.5 in. Some furnace-men maintain that the hole ought to be 4 in. in diameter so as to permit flushing out mush which is not taken up by slag or matte. If this mush accumulates, it may close up the first and even the second tuyères near the front. The 4-in. tap-hole is usually opened only in part, but is very serviceable when occasions demand a thorough cleansing.

Soft-steel Jackets.—The general features of the steel jackets enclosing the smelting zone are the same as those forming the shaft-walls. They are usually higher than those of cast-iron; thus Figs. 174 and 175 show jackets 6 ft. high, and Figs. 176 and 177, 180 and 181, 184 and 185, jackets that are 6 ft. 6 in. high, whereas with cast-iron the usual height does not exceed 4 ft. The width of the lower steel jacket is the same as that of the upper, as are the water-space and the thickness of plate. One characteristic, not shown in the drawings, is that the bottom of a jacket is not horizontal, but inclined outward so

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that mud and scale are not distributed along the bottom, but collect in a trough formed by the inclined bottom and the outer sheet, whence it is removed periodically through hand-holes. The lower jackets are joined and held in place in the same manner as are the upper. Both have water-inlets and outlets near bottom and top. The usual form of outlet is shown in the Holthoff jacket, Fig. 199, in which stress is laid upon giving the curvatures, *A* and *B*, at corners large radii, especially on the fire-sheet, so that they take up readily the expansion caused by heating.

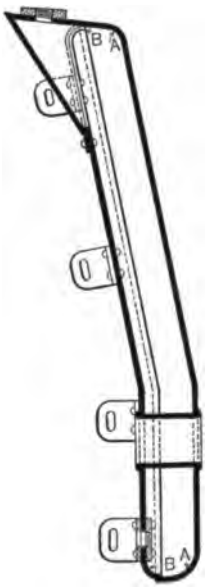

FIG. 199.—Holthoff steel jacket.

The lower steel jackets either have a bosh, Figs. 174, 177, 180, as do the cast iron, Fig. 173, or not, Fig. 184. The former is more common; with charges made up mostly of blast roasted material the bosh is losing its former importance.

The cooling-water for the jackets is drawn from a main tank placed some distance above the water-inlets of the jackets in order that the water may have the head necessary to force through the pipes the extra amount of water needed over the normal, when the furnace is blown in or out. The positions of the main and branch pipes are shown in the drawings. The amounts of water used in modern furnaces are given in Table 53. The jackets are washed out at intervals to remove accumulating mud. This is done more frequently with the lower than the upper. With clean water that is not hard, the upper jackets are washed out perhaps once a month or oftener when it becomes necessary to bar off hangings.

In many instances when there is a lack of water, the same water is used over and over. This necessitates cooling apparatus, of which there exist many forms.[1]

112. Tuyères.—The diameter of tuyère-opening is 4–4½ in.; the center is placed usually about 13 in. above the bottom of the jacket; and the distance between centers is 16–18 in. The diameter used to be 3 in.; the increase to 4 and 4½ in. is due to the desire of increasing the smelting power, and has been made possible by the comparatively small amount of fines in the charges treated at present, which are made up largely of blast roasted materials. Before the advent of blast roasting, fine ores formed a large part of the charge; this caused more or less tightness and offered a much greater resistance to the passage of the blast than does the coarse charge prevailing at present.

[1] Henrich, *Tr. A. I. M. E.*, 1895, XXV, 43, 460.
Schmitt, *Proc. South Afric. Assoc. Eng.*, 1906–07, XII, 25.
Hart, *Eng. Mag.*, 1907–08, XXXIV, 590.
Roberts, *Jl. South Afric. Assoc. Eng.*, 1909, XV, 33.
Patteson, *Cass. Mag.*, 1909, XXXV, 701.
Jawger, *Min. Sc.*, 1911, XLIV, 480, 495.
Wheeler-Balcke, *Iron Age*, 1913, XCI, 646, *Eng. Min. J.*, 1913, XCV, 1051.

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Too much emphasis has been laid upon the jet-like action or penetration of the blast and too little upon the volume. At Great Falls, Mont.,[1] Wheeler has shown that volume of air is the essential feature in governing the smelting power of the furnace, and that penetration has comparatively little to do with it.

It has been held that penetration is a function of the diameter of tuyère-nozzle. Austin[2] has opposed this idea. More recently measurements, made by Towmay and Hinde[3] at the Cockle Creek Works, N. S. W., have shown that with a pressure of 46 oz. per sq. in. in the blast main, the pressure in a 3-in. tuyère was 40–41 oz., and that in the furnace, 6 in. beyond the tuyère, it ranged from 35–41 oz. Reducing the 3-in. tuyère to 2 in. showed that for the same volume of blast the pressure in the main had to be raised from 46 to 51 oz. A small tuyère requires therefore much more power for the same volume of air than does a large. The air upon leaving the tuyère travels in large quantities through open channels and in relatively small amounts through spaces where it meets resistance from fine materials. Of course, there must be some pressure to overcome the resistance in the smelting zone; it is usually from 2–3 oz. and may reach 5–6 oz. below the pressure at tuyère, a difference ample to permit the air, necessary for smelting, to pass into the furnace.

The number of tuyères desired for a given hearth area has undergone some change. Hahn[4] reckoned in 1882 that one 3-in. tuyère could serve 2 sq. ft. of hearth area. This corresponds to a tuyère-ratio[5] of 3.53; in Table 53 it will be seen that the ratio with most furnaces in operation at present is about 5.

The distance of tuyère-center above bottom of jacket used to be 10 in., and is now 13 in. The former figure was sufficient for the small amounts of matte produced in smelting mixed oxide and roasted sulphide ores. With the increase of matte-fall the height had to be raised. If double treatment in blast roasting can reduce the S-content to 2 per cent. the distance may be reduced again. The nearer the tuyère-level is to the top of the crucible, the hotter will be the slag and matte, and thereby also the lead in the crucible.

Tuyères are placed usually on the sides of the furnace, with centers 15–18 in. apart in steel jackets; with cast iron the distance was 20 in. The more closely the tuyères are placed, the smaller will be the dead places in the furnace. With tuyères on the sides only, there is danger of crusting at the back; with a back-tuyère of the same size as the adjoining side-tuyères, the latter will grow cold. W. Wraith[6] using at Tooele, Figs. 174 and 175, 4-in. tuyères, 12 to a side and one at the back, by means of a bushing reduced the diameters of the tuyères adjoining the back-tuyère to 2 in., and obtained a uniform heat.

The tuyère-pipes in use at present[7] are very different from those 15 years

[1] Church, *Tr. A. I. M. E.*, 1913, XLVI, 437.
[2] *Tr. A. I. M. E.*, 1898, XXVIII, 902.
[3] *Eng. Assoc. of N. S. W.*, through *Eng. Min. J.*, 1916, CII, 392.
[4] *Min. Res. U. S.*, 1882, 336.
[5] Sq. in. tuyère area : sq. ft. hearth area.
[6] Private notes, 1916.
[7] Vail, *Eng. Min. J.*, 1916, CII, 639.

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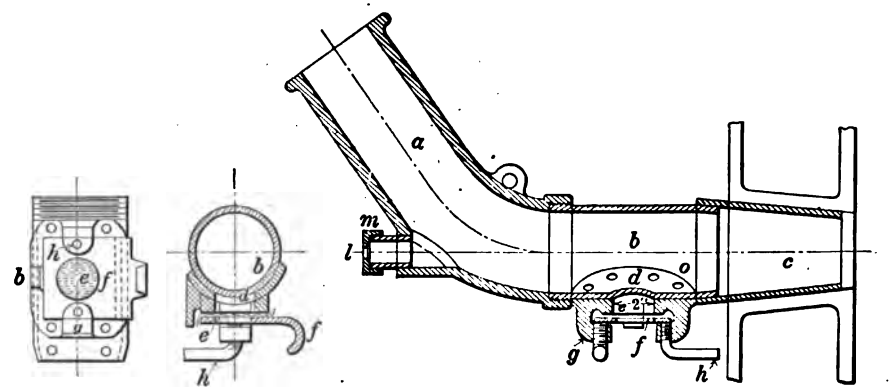
ago although the general form has remained the same. The older form is shown in Figs. 172 and 173. The pipe is a tube of galvanized iron; the horizontal tube, about 14 in. long, fits into brass nozzle, p', inserted into the tuyère-hole, or soldered to it; elbow, x, which connects the horizontal arm with tuyère-bag, q, has a brass nipple which contains the eye or peep-hole, s, closed either by a slide or cap having a glass or mica plate in the center, or simply by a wooden plug. In the center of the plug is left a small opening, the size of a pencil, to be closed by a piece of wood. In order to keep the blast-pipe in its normal position and thus to prevent it from delivering the blast upward, which is its natural tendency, an iron band, hooked by means of two springs to the jacket, is passed around the elbow; or an iron loop is soldered to the inner side and hooked to the jacket.



FIGS. 200 to 202.—Tuyère-pipe with Davis slag-escape.

The details of an improved form of such a tuyère-pipe with the Davis slag-escape[1] are shown in Figs. 200–202, in which a is a cast-iron elbow, with peep-hole, l, closed by cap, m, screwed to a 4-in. wrought-iron pipe, b, provided with opening, d, for the escape of slag, and soldered to conical brass nozzle, c, entering the cast-iron jacket. To the under side of b is riveted flange, d, with lugs, g. They receive in a recess the perforated disc of sheet-iron, f, carrying a piece of paper or sheet lead, e, and pressed against the seat by screws, h, thus making an air-tight joint. If slag runs into the tuyère, it will burn the paper or fuse the lead, and the noise of the escaping blast calls the attention of the keeper to the accident.

With the ordinary pipe inserted into the tuyère-hole, a leakage of air can not be prevented, even if all parts are carefully fitted. The loss in air became excessive when the blast-pressure was raised from 12 to 48 oz. and over to meet the needs of wider and higher furnaces. This led to the adoption of a tuyère-box fastened by an air-tight joint to the jacket. Transition forms, such as those of Austin,[2] Unziger, and older forms of tuyère-boxes, are given in the preceding edition of this book.

[1] Hofman, *Eng. Min. J.*, 1893, LVI, 396.

[2] *Min. Sc. Press*, 1909, XCVIII, 392.

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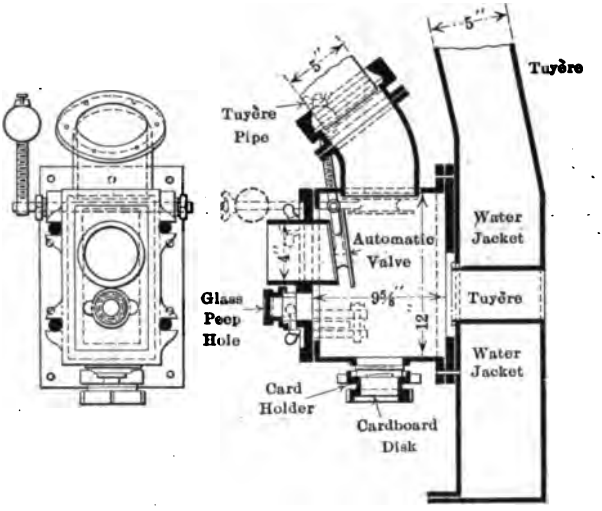
Examples of modern tuyères are those of Eilers, Anderson, and the Traylor Engineering and Manufacturing Co.

The Eilers tuyère, used by many plants of the American Smelting and Refining Co., is shown in Figs. 203 and 204. It consists of a cast-iron box screwed tightly to the jacket. On the upper side is a nipple with flange to be connected with a flanged 5-in. tuyère-pipe; on the lower side is a 5-in. slag-escape closed by cardboard kept in place by a holder; in the cover, attached to the rear end by hinged bolts, are two openings, the glass-covered peep-hole, and the 4-in. outlet for the escape of the gases when the blast is shut off. While the blast is on, the automatic valve, provided with adjustable counter-weight, is pressed by the blast against the seat of the 4-in. escape; when the blast is off, the valve



FIGS. 203 and 204.—Eilers automatic cut-off tuyère.

closes the blast-inlet and opens the outlet, through which the back-pressure in the furnace forces the gases into the open. The tuyère is excellent when everything is working normally; troubles arise when dirt or slag gets in accidentally, causing leaks or warping the disk.

The Messiter tuyère[1] has another device to prevent the back-flow of furnace-gases into the tuyère-pipe.

The Anderson tuyère, used by the U. S. Smelting, Refining, and Mining Co., at Midvale, Utah, is shown in Figs. 205–212. The cast-iron tuyère-box, clamped to the jackets, has an 8-in. socket to receive the ball of the sheet-iron tuyère-pipe, a slag-outlet closed by a nipple, cap, and lead washer, and a tuyère-box cap with lugs tightened by means of cap screws.

The Traylor tuyère, used at Trail, Northport, and Kellogg, is shown in Fig. 213. This combines several interesting features. The cast-iron tuyère-box

[1] U. S. Patent, No. 706332, Aug. 5, 1902.

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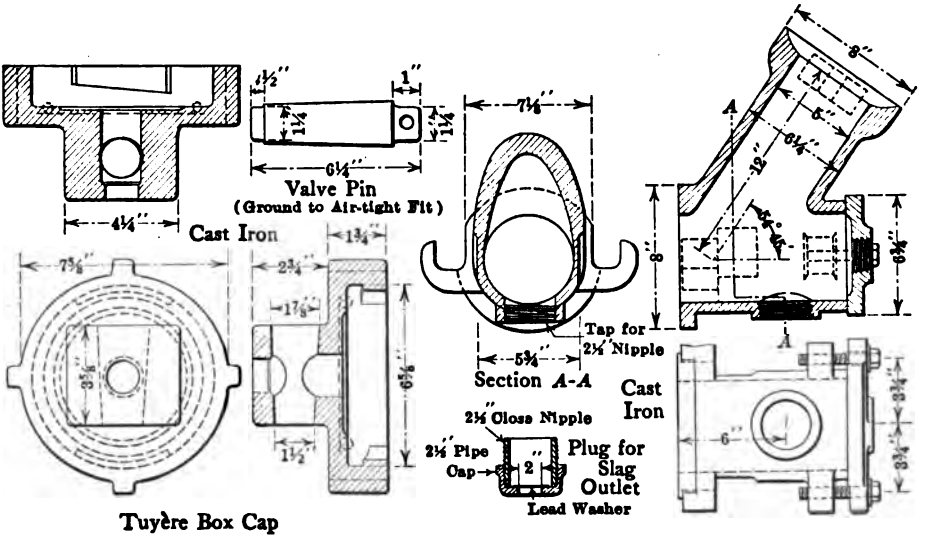
is clamped to the jackets by means of hooks fastening in open lugs; the hooks threaded at one end are tightened by means of nuts, and the joint between box



FIGS. 205 to 212.—Anderson tuyère.

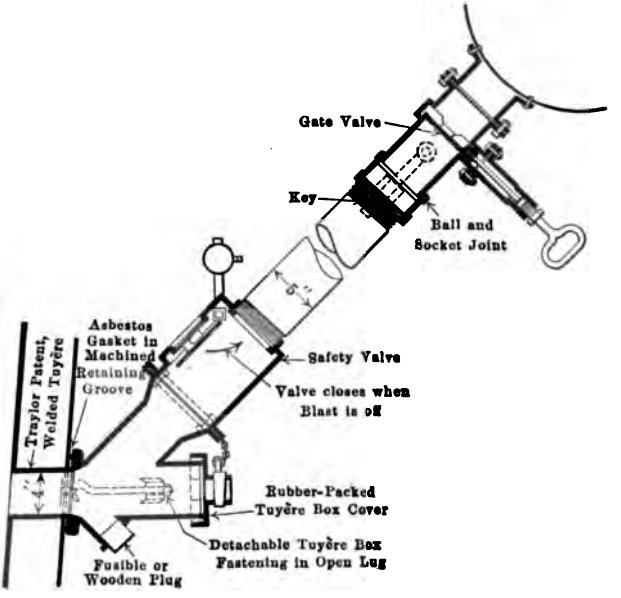

Gate Valve
Key
Ball and Socket Joint
Asbestos Gasket in Machined Retaining Groove
Traylor Patent, Welded Tuyère
Safety Valve
Valve closes when Blast is off
Rubber-Packed Tuyère Box Cover
Detachable Tuyère Box Fastening in Open Lug
Fusible or Wooden Plug

FIG. 213.—Traylor tuyère.

and jacket is made air-tight through an asbestos gasket held in a groove of the box. The box has the Robinson slag-escape[1] closed by a fusible or a wooden

[1] *Eng. Min. J.*, 1908, LXXXV, 251, LXXXVI, 756.

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plug, and a rubber-packed cover provided, as in the Anderson tuyère, with a valve-pin or tapered plug in the peep-hole. The tuyère-box proper is followed by a casing containing a safety-valve similar to that of the Eilers tuyère. The 5-in. sheet-iron pipe is connected by ball- and socket-joint to the casting containing the gate-valve, and this is joined with the bustle-pipe as shown in the drawing.

113. Blast.—The air blown into the furnace is not pre-heated. The use of hot blast has been repeatedly suggested. Experiments made years ago[1] showed no saving in fuel; in fact the burning of extraneous fuel to pre-heat the air for the blast furnace can not be considered as promising any economic results. Utilization of the waste heat of slag will cause a saving in fuel, but the question, unanswered so far, is whether the cost of the additional plant is not greater than the amount of coke saved.

The apparatus in general use for furnishing blast is the rotary pressure blower of the Roots type. In a few cases turbo blowers have been installed; they occupy a much smaller space than do rotary pressure blowers, work very smoothly, and, it is believed, also very efficiently, although serious doubts have been expressed upon this point.[2] Tests made with Connersville blowers at Midvale, Utah, and Trail, B. C., show a blast efficiency of 85 per cent. of the indicated horsepower when the blower is working against a pressure 2.5–3 lb. per sq. in. The volume of air furnished by rotary pressure blowers is large; the work is satisfactory as long as a furnace does not require a pressure higher than 3 lb. per sq. in., otherwise they would not be in general use with non-ferrous blast furnaces. Table 53 shows that modern blast furnaces take from 4,000 to 8,000 cu. ft. per min. This calls for large blowers. Tables 55 and 56 give the principal facts of the two leading forms of rotary pressure blowers.

TABLE 55.—ROOTS ROTARY PRESSURE BLOWER

No.	Displacement per rev., cu. ft. air	Rev. per min.	H.p. per 1,000 cu. ft. displacement per min. at 2.5 lb. pressure	Discharge diam., in.	Pulley		Machine	
					Diam., in.	Face, in.	Length by width by height, in.	Weight, lb.
4	12.2	270	41	14	32	6	97 7/16 × 41 3/16 × 42 1/16	5,000
5	18.27	230	51	16	36	8	115 5/16 × 49 × 48 1/16	8,500
5½	26.1	200	62	18	42	10	126 1/16 × 54 3/16 × 54 1/4	10,000
6	35.7	190	80	20	48	12	140 3/16 × 59 × 57	13,500
6½	47.5	175	95	22	66	12	159 5/16 × 65 3/16 × 63 3/4	16,000
7	56.9	160	110	24	70	14	168 1/4 × 73 × 55 3/4	26,000
7½	81.0	145	140	24	84	16	185 × 79 3/16 × 62	36,000
8	111.2	135	175	30	106	18	202 3/4 × 90 × 67 1/4	40,000
8½	147.9	125	220	30	120	22	220 1/16 × 100 × 77 1/8	52,000
9	192.0	120	260	36	136	24	.....	70,000
9½	244.0	110	320	36	160	26	.....	85,000
10	304.9	100	360	42	184	28	.....	115,000
11	401.0	95	450	42	216	32	.....	130,000

[1] Merbach, "Die Anwendung der erwärmten Gebläseluft im Gebiete der Metallurgie," Leipsic, 1840, pp. 157–166, 168–170, 178–180.

Le Chatelier, *Ann. Min.*, 1840, XVII, 3.

Wedding, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1871, XIX, 155.

[2] *Comp. Air Mag.*, 1916, XXXI, 7952.

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TABLE 56.—CONNERSVILLE ROTARY BLOWER

Displacement per rev., cu. ft. air	Rev. per min.	Discharge diam., in.	Pulley		Machine, length by width by height, in.	Power
			Diam., in.	Face, in.		
13	250	14	42	7	99¾ × 45½ × 39	5 h.p. per 1,000 cu. ft. displacement per min. at 1 lb. pressure
17	225	16	42	8	117 × 48 × 41	
24	200	18	48	10	123 × 54½ × 44	
33	190	20	60	12	147 × 60 × 49½	
45	180	20	66	14	160¼ × 65½ × 55	
57	170	24	72	16	170¾ × 70½ × 59	
65	160	24	84	16	181 × 76 × 63	
84	150	27	84	20	192 × 81½ × 67	
100	140	30	96	20	210½ × 87½ × 70	
118	130	30	120	20	213¾ × 94 × 76	

There are two ways of supplying blast to a number of furnaces. Each furnace can have its own small-size blower, or several large-size blowers deliver the compressed air into a blast-main from which the several furnaces are supplied. The first method is the more desirable, as a furnace can be run independently from its neighbors, while at the same time it can be cross-connected with a main leading to other furnaces and assist in time of need. The second is the one generally accepted, as the plant is cheaper, the care easier, and the repair smaller; it is, however, desirable that the blowers be of the same size. The volume of air delivered to the several furnaces with the second method is regulated by a gate in the branch-pipe.

The blast-mains are of plate iron. The main blast-pipe or induction-pipe which receives the air from several blowers has a diameter one-third larger than the combined outlets of the blowers. The main usually runs along the backs of a row of furnaces near the dust-chamber, and is suspended 8 ft. or more above the furnace floor. It has safety- or relief-valves and is closed at both ends by gate-valves. If one of the furnaces is to stop for a short time, one of the gate-valves is slightly opened and some air allowed to go to waste; this is a more convenient procedure than reducing the speed of the engine. In each branch-pipe connecting a blower with the main pipe there is a gate which is closed when the blower is not in use, as otherwise it may happen that the idle blower will be running backward.

From the blast-main branch-pipes, each provided with a gate-valve, furnish the blast to the bustle-pipes of the several furnaces, Figs. 176 and 181. The bustle-pipe, which encloses two sides and one end of a furnace, Fig. 214, has on the upper surface cast-iron gas-vents, the details of which are shown in Figs. 215 and 216. The opening of the vent, 10 in. in diameter, has bolted to it the valve-seat, through the center of which passes a gliding brass valve-rod with handle carrying at its lower end the valve, a brass plate. The rod fits snugly into the opening of a rubber washer. The combined weights of valve and rod are so adjusted that the valve, when pulled with the handle, will remain in place when the blast-pressure is normal; it will sink when the pressure falls.

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On the lower side of a bustle-pipe thimbles pass downward to be connected with the tuyères. Formerly the connection was made with closely woven canvas bags, Fig. 173, which had been soaked in water-glass, alum, or mineral paint to prevent catching fire from sparks. A thimble had a slide-damper to permit shutting off the blast, wholly or partly, from a tuyère.

With high-pressure blast the leakage became so great that the bag was replaced by an iron pipe provided with a gate-valve.

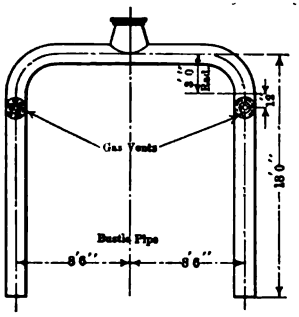

FIG. 214

FIG. 214.—Bustle-pipe with gas-vents.

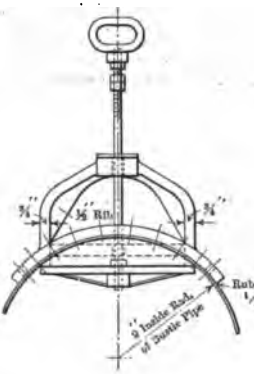

FIG. 215

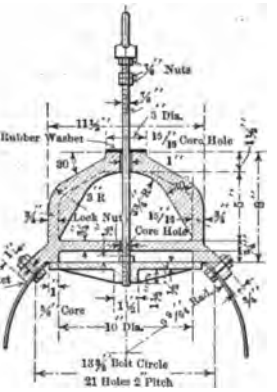

FIG. 216

FIGS. 215 and 216.—Gas-vents of bustle-pipe.

The pressure of the blast is measured with quicksilver, water, or glycerine gauges; a difference in level of 2 in. quicksilver or 28 in. water or 21.5 in. glycerine (sp. gr. 1.28) equals 1 lb. or 16 oz. av. d. p. pressure per sq. in. Water and glycerine are often colored, or a drop of insoluble coloring matter is added to each limb that the level may be plainly seen. Most works at present have in the blower room a recording gage which shows the variations in pressure of blast in 24 hr. The recorder of the Bristol Co., Waterbury, Conn., is the one most frequently found.

114. Handling Ores. Fluxes and Fuels.—The receiving and disposing of the raw materials for the blast furnace forms part of the general arrangement of the whole plant with its operations of sampling and storing of materials, of roasting and blast roasting, preparing smelting charges and disposing of products, as well as the disposition of power, water, and other necessaries. The correct solution of this engineering problem may involve the success or failure of a plant, and is too large to be taken up here in detail.

There have taken place many discussions as to whether a terraced or a level site[1] is best suited for a plant, but the views expressed differ greatly, although

[1] Locke, "Smelting Plants," Cincinnati, 1883.
Austin, *Tr. A. I. M. E.*, 1896, XXVI, 388.
Vezin, *ibid.*, 1095; *Eng. Min. J.*, 1897, LXIV, 123.
Editor, *Min. Reporter*, 1904, L, 376.
Lang, *Eng. Min. J.*, 1907, LXXXIII, 565.
Austin, *ibid.*, 726.
Moore, *loc. cit.*, 1908, LXXXV, 546.

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everybody agrees upon the necessity of having ample fall and space for the slag-dump. The handling of materials by belt conveyors has weakened the case of the advocates of a terraced site.

In the preceding edition of this book drawings and descriptions of several well-arranged plants were given. Some of these plants have been abandoned; others have been altered to meet the present requirements of mechanical handling of materials in the preparation of charges and the disposal of products. The modernized works cannot serve as models. The only new plant is that of the Bunker Hill and Sullivan Mining and Concentrating Co. at Kellogg, Idaho;[1] another has been proposed for China, but not erected; both are treated in §122.

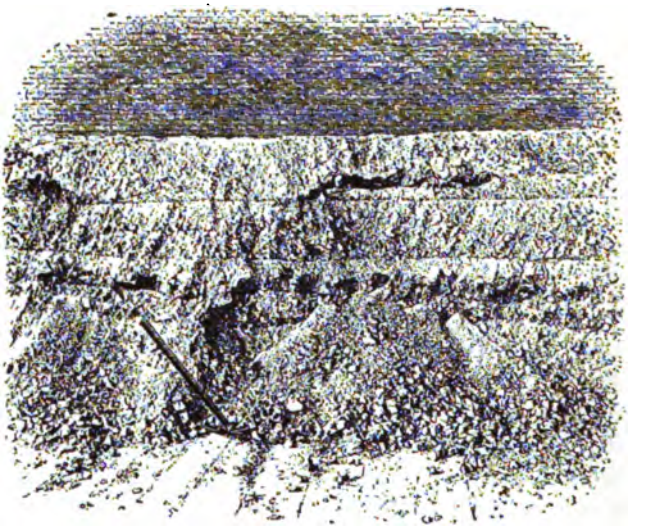

FIG. 217.—Ore-bed.

Some plants situated in the North have to handle in winter frozen concentrates. These have been thawed out, *e.g.*, on a straight-line, heated hearth over which travel scrapers. This method is expensive and is likely to make much dust, especially with flotation concentrates. Crushing the frozen ore with toothed rolls and then reducing to desired size with a swing-hammer pulverizer is more satisfactory, especially as flotation concentrates form granules which give up their water only when they have been fed on to the Dwight-Lloyd sintering machine, and are not handled before they have been sintered with the charge to which they had been added.

In order to make possible a general discussion of the handling of raw materials, it is expedient to consider with some details, the methods followed by the older and the remodeled plants in receiving the raw materials and preparing the charges with manual labor and mechanical devices.

[1] *Eng. Min. J.*, 1916, CI, 868; *Min. Sc. Press*, 1917, CXIV, 155.

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A custom lead smeltery[1] receives a variety of ores which, after having been weighed, sampled, and analyzed separately, have to be incorporated in the smelting mixture. This has always been, and is still largely accomplished by bedding, that is, spreading out the ores in horizontal layers (similar to layer-cake), calculating the average compositions of the beds, and using as ore mixtures the beds which are cut down vertically so that each cut shall represent a true average. A section of a bed is shown in Fig. 217.

In making up beds, the physical conditions as well as the chemical compositions of the ores are taken into consideration. Thus physically, ores may be coarse or fine, dry or wet; chemically they may be silicious, basic, or nearly neutral, they may be sulphides which are to be roasted or blast roasted, or oxides which are smelted raw. Analyses of such ore-beds at Monterey, Mexico, by Hahn[2] are given in Table 57.

TABLE 57.—ANALYSES OF ORE-BEDS

	SiO_2, per cent.	Fe, per cent.	CaO, per cent.	Zn, per cent.	Al_2O_3, per cent.	S, per cent.	As, per cent.	Pb, per cent.	Ag, oz. per ton.	H_2O, per cent.
Lead ores	13.6	21.6	3.3	2.3	.....	1.5	.....	27.0	15.4	12
	19.2	19.3	4.2	3.0	.....	3.0	.....	21.1	26.9	12
	6.5	28.5	2.0	1.2	.....	1.0	.....	18.7	7.8	12
Iron ores	5.5	40.5	2.8	2.6	.....	1.2	.....	8.1	2.5	8
	6.6	35.5	3.0	1.5	.. ...	1.2	.....	9.0	2.3	8
Silver ores	35.5	7.0	17.8	1.0	.....	4.8	1.0	.....	53.6	8
	44.2	4.6	14.7	2.0	.....	2.4	.....	.....	263.3	8
	73.4	3.5	6.8	1.5	.....	2.3	.....	.....	30.0	.....
Special ores	15.6	35.9	1.0	6.8	1.7	8.2	.....	.....	32.1	5
	14.3	37.9	0.9	5.6	.....	6.2	.....	1.0	35.9	6

With large smelting-beds it is frequently not possible for small shipments to be spread over the entire bed; in such cases small lots are "matched," that is, they are distributed in parallel strips over the ore-bed.

In modern mechanical plants the practice of bedding has been mostly abandoned, although there is a prospect that the Dwight-Messiter[3] system, which has been so successful with copper smelteries, will make its way into lead plants.

The following examples of the preparation of ore mixtures are characteristic of the present modes of operating.

[1] Editor, *Eng. Min. J.*, 1912, XLIV, 50.
Eilers, *ibid.*, 147.
[2] *Tr. Instit. Min. Met.*, 1899–1900, VIII, 262.
[3] See Hofman, "Metallurgy of Copper," McGraw-Hill Book Co., New York, 1918, 356.

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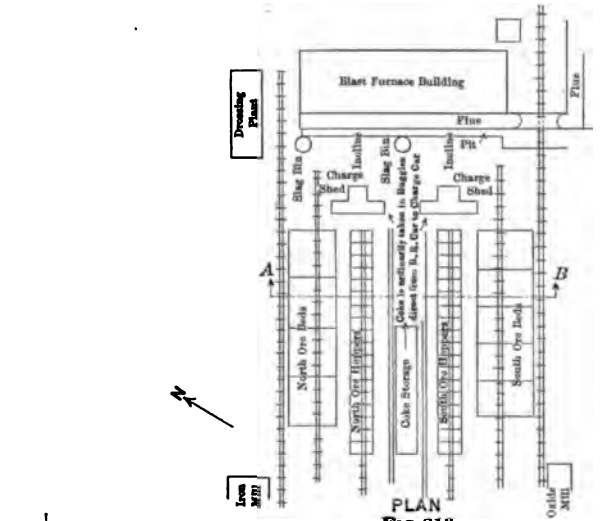
Plant A.—In Figs. 218 and 219 are shown the plan and cross-section of bedding-bins and ore-pockets of a large smeltery; the section is drawn to a larger scale than the plan in order to bring out details. The plant is 335 ft. wide and 512 ft. long, and covers an area of 171,520 sq. ft. For the reception of the materials there are provided the North and South ore-beds, and the North and South ore-hoppers or -pockets on the sides of the central coke-bins; usually the coke is unloaded from the cars directly into the buggies. The materials arrive in cars on elevated tracks, and are discharged either by hand from the side of a car or dropped from the bottom.



PLAN
FIG. 218

SECTION *A-B*
FIG. 219

FIGS. 218 and 219.—Plant A; plan and cross-section of bedding-bins and ore-pockets.

For the bedding of ores there are provided two sets of four bins, 16 by 20 ft. and 16 ft. 6⅜ in. high, each holding about 50 tons of ore. A set receives its ore through two elevated tracks. The ore is sampled by fractional selection; the discard is transferred by wheelbarrows from the cars to the bins and spread. Ore and flux to be stored in the hopper-shaped bins arrive as a rule in bottom-discharge cars on elevated tracks. There is provided a large bin for storage of coke which is to supply the furnaces in case the daily shipments cease temporarily; usually the coke-cars are emptied into wheelbarrows. Ore, flux and coke are wheeled to the charge-shed and emptied there into a charge-car on a depressed track in the charging pit, care being taken to dump the charge-components in prescribed order in such a manner that they form layers; coke is introduced last in order to diminish breaking by the impact of the charge, which

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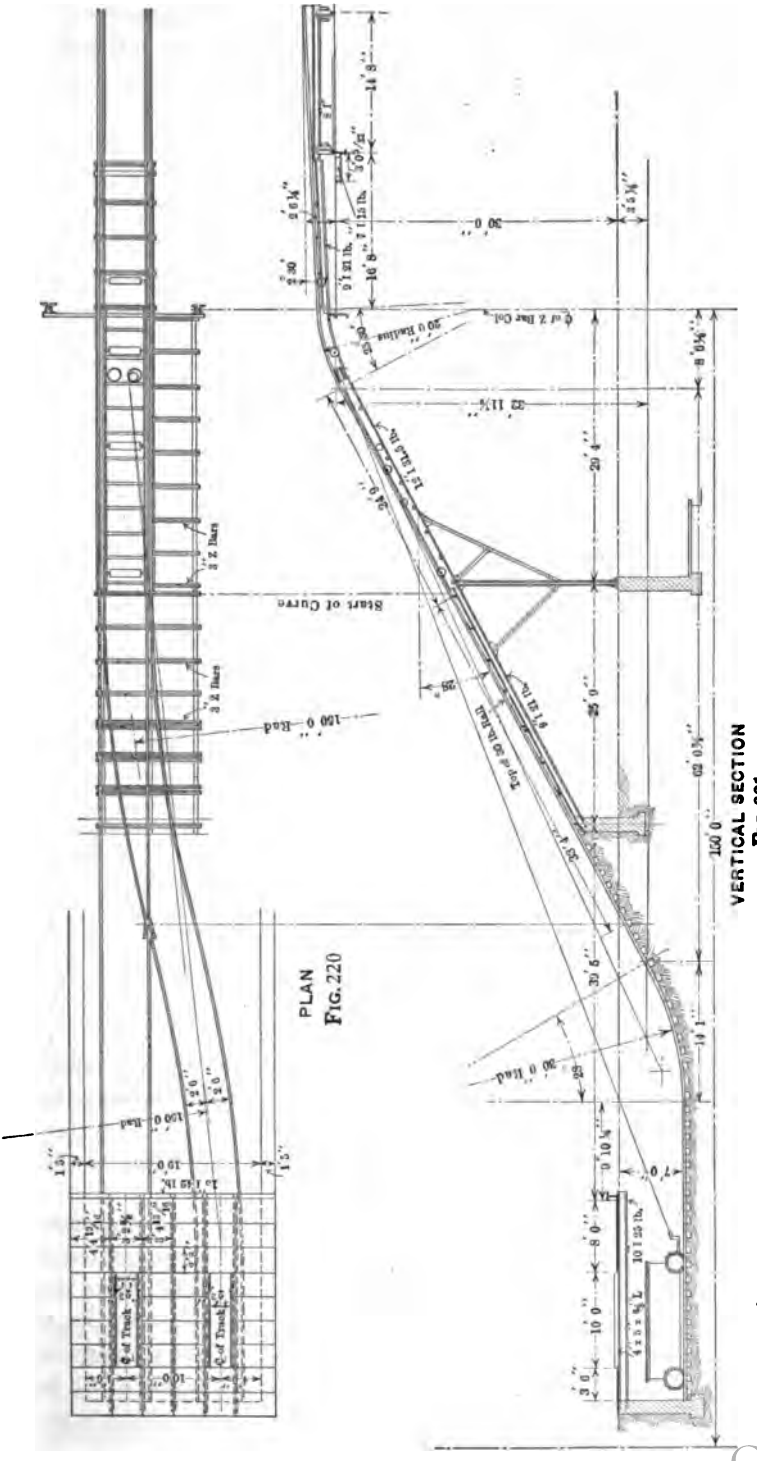

FIGS. 220 and 221.—Incline for blast furnace charging car.

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weighs 8,000 lb. An inclined hoist delivers the bottom-discharge car to a transfer-car on the feed-floor, which is moved to and fro by electric traction, spotted over the throat of a furnace, whereupon the contents of the charge-car are dropped.

Details of this arrangement are given in plan and vertical section in Figs. 220 and 221. The plan shows in the iron-plate platform the openings, 10 ft. by 2 ft. 4⅜ in., of the two charging-pits in the charging-shed of Fig. 218. The openings have on the sides L-shaped angles, 4 by 5 by ⅜ in., against which the hand-pushed cars bump when they are being discharged. Their contents are received in two charge-cars, 11 ft. 9 in. by 3 ft. and 3 ft. 6 in. deep, traveling on a track of 3-ft. gauge; a car holds 5–6 tons charge. The cars, 7 ft. below the floor, are raised 30 ft. by a direct-hoisting cable to the feed-floor on two inclines,

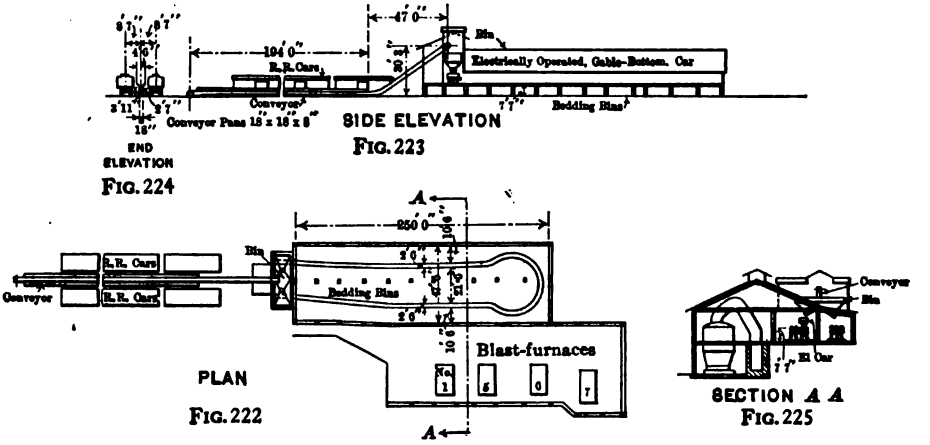

FIG. 222–225.—Unloading and bedding direct ore.

of 28° grade, which combine to a single line leading at the crest of the incline to the transfer-car on the feed-floor which has a gauge of 14 ft. 8 in. When the charge-car has been unhooked, it is transferred to a furnace, emptied, and returned to the hoist; the hoisting-cable is hooked on, and the car lowered into the charging-pit. A switch at the upper end of the incline deflects the car to its proper track.

This method of raising the charges to the feed-floor is similar to the one in operation at East Helena, Mont.[1]

Plant B.—A combination of bedding ores mechanically and transferring the mixture by hand to the feed-floor of the blast furnaces is represented by plan, side- and end-elevations, and section in Figs. 222–225. The direct ores, that is, ores to be bedded and smelted raw, arrive in box cars on two tracks; they are unloaded by hand from the platforms, 3 ft. 3 in. high, onto a chain-conveyor, 2 ft. 7 in. high, running between the platforms. The conveyor, which has pans

[1] Dwight, *Tr. A. I. M. E.*, 1902, XXXII, 382

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18 by 18 by 8 in., travels horizontally for a distance 194 ft., rises 30 ft. 3 in. in a distance of 47 ft., and delivers the ore into a collecting bin provided with gable-shaped bottom and corresponding double discharges which empty into side-discharge cars. At an elevation of 7 ft. 7 in., these are hauled electrically outward

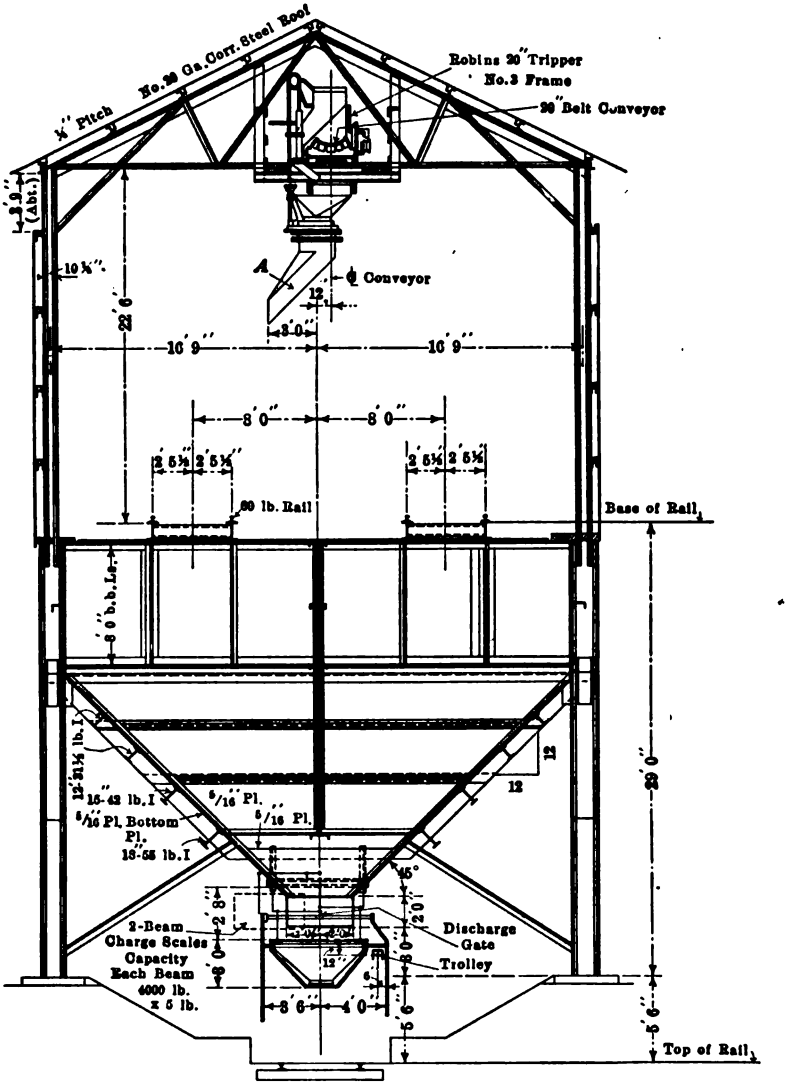

FIG. 226.—Section of storage- and bedding-bins, Tooele, Utah.

on one track and backward, after passing the loop, on the other; the gauge is 2 ft. 6 in. The man on the cars attends to the starting, stopping, and discharging. The bedded material is moved by hand in trucks and wheelbarrows to the feed-floor of the blast furnaces, which is on the same level.

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Both arrangements provide for the bedding of ores, but involve considerable manual labor, plant *A* more so than plant *B*. The present tendency of lead plants is to give up the bedding-system, to handle ores mechanically, and to make up charge-mixtures in a similar manner.

Plant C, shown by a section in Fig. 226, represents the storage- and bedding-bins of the lead-plant of the International Smelting Co., Tooele, Utah.[1]

The series of 26 V-shaped bins is 260 ft. long and 32 ft. wide. At first a V-shaped bin was divided by a central wall into two compartments; there were in operation 52 bins which were 16 ft. long. As the ores dropping from opposite bins into the cars below were not satisfactorily mixed, the central wall has been removed and only the steel framing retained. There are in use therefore 26 bins which at the base of the V are 32 ft. long, 10 ft. wide, and have a maximum depth of 24 ft.; a bin or pocket has a capacity of 5,398 cu. ft. and holds about 500 tons lead ore. The several pockets receive the ores either direct from railroad cars, or through a 20-in. belt conveyor, 820 ft. long, and a Robins automatic traveling tripper. Thus fine ores and concentrates are usually unloaded direct from cars, whereas other materials go first to the crushing plant whence they arrive by the belt conveyor. A pocket is closed by a circular hand-operated gate. The ore is discharged into a suspended weighing-hopper provided with Fairbanks suspended scales having independent beams. The weighed materials of the hoppers are emptied into the charge-cars running slowly to and fro on the bottom track, so that they are uniformly spread in the cars. Ore that is to go to the Dwight-Lloyd sintering machines is discharged into cars 12 ft. long and 4 ft. wide holding from 3 to 5 tons; material which goes to the blast furnaces is collected in hopper-bottom charge-cars having two drops which are 14 ft. long and 5 ft. wide, and hold from 5 to 8 tons. A charge-car has four-wheel trucks at each end, and is moved by a railway-type motor geared direct to the axle. A car thus has two motors which are operated at one end through a street-railway type controller. The drop-doors and brake mechanism can be operated from either end of the car. The charges for the five charge-cars of the five blast furnaces are made up per shift by 1 foreman, 3 loading men, 2 motor men, and 1 laborer who shovels in drosses. On the furnace-floor are 1 foreman, 2 feeders, 2 helpers, 1 laborer, and 1 coke-weigher. Thus 14 men handle per shift from 1,300 to 1,580 tons charge.

Plant D.—A cross-section showing one of the concrete bins, the tilting weighing-hoppers and receiving charge-cars of the Consolidated Mining and Smelting Co. at Trail, B. C., is shown in Figs. 227 and 228. The six elevated concrete bins of a section receive their materials from a track not shown; they have convex fronts and backs and parallel sides, and each has three discharges 2½ ft. wide. A bin, 18 ft. long, 16 ft. 6 in. wide, and 14 ft. deep, has, without the discharge, a capacity of 4,358 cu. ft. and holds about 450 tons lead ore. The discharge-openings, 16 by 24 in., are partly closed by gates. The ore is fed from a bin by link-belt conveyors having metal curtains. A conveyor, when

[1] Thomson-Sicka, *Tr. A. I. M. E.*, 1913, XLVI, 946. Private notes, and communication by W. Wraith, July, 1916.

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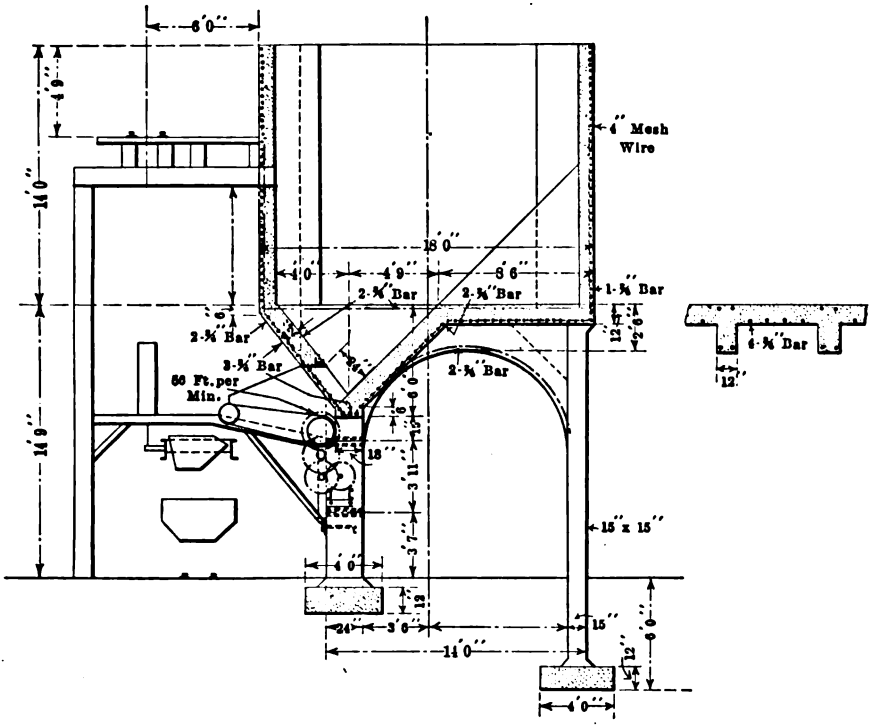

FIGS. 227 and 228.—Charge-bin and weighing-hopper, Trail, B. C.

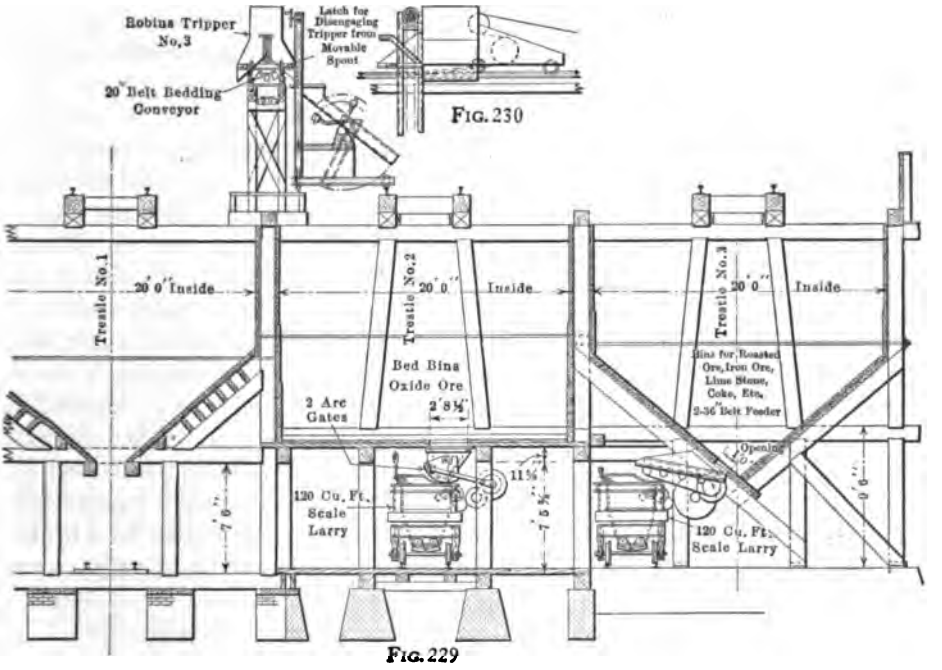

FIG. 229

FIGS. 229 and 230.—Charge-bin and scale-lorrie, Midvale, Utah.

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in motion at a speed of 50 ft. per min., delivers the ore into a tilting receiver provided with a beam scale. The receiver empties automatically into the charge-car beneath when it has received its predetermined amount of ore.

Plant E.—A very recent installation is that of the United States Smelting, Refining, and Mining Co. at Midvale, Utah,[1] which has reduced the number of men required to a very low figure. Fig. 229 gives a cross-section through the ore-bins, and Fig. 230 a side-view of the Robins tripper. The materials are collected in three rows of bins, each with its track beneath. On the tracks, connected by spring-switches, travel the scale- (charge-collecting) cars. One row of bins has flat bottoms; they receive sampled oxide ores to be bedded, and are provided with pairs of reciprocating undercut arc-gates. The bedded ore is cut down by two men in a bin and shoveled to the arc-gate. The oxide ores are distributed on the beds by means of a slow-moving Robins tripper which discharges from a belt conveyor. To the standard is attached a carriage which, running on a track below the level of the belt conveyor, has an adjustable spout to permit delivering the ore to the near or far side of the bin. The tripper proper has mounted on it a seat for the operator who starts and stops the tripper, and adjusts the movable spout. The other two rows have hopper-bottom bins for roasted ore, iron ore, limestone, coke, etc. A bin is discharged by a pair of (Leviathan) belts. The discharges of the bins are wide, the combined width being a little less than the length of the charge-car; they therefore fill the car uniformly throughout its length. The discharge-gates are operated by a 7.5-h.p. motor on the car. The car has about the same length as that of the throat of the blast furnace; its capacity is 120 cu. ft. It is mounted on two 4-wheel swivel-trucks and one pair of small guide-wheels, which permits running on sharp curves. The car carries a 19-h.p., 500-volt direct-current travel motor, which drives it with full load at a speed of from 600 to 700 ft. per min. There are provided in addition a 7.5-h.p. motor for opening and closing the gates, and a foot-brake. The hopper-frame of the car is suspended on scales having multiple beams. The scale-pointer is provided with a plus- and minus-beam which is set before every weighing in such a manner that the pointer moves when the weight is 100 lb. short of the final weight. By setting the auxiliary beam on the zero mark, the weigher can add the missing amount and obtain an accurate weight. In operating, the car is stopped beneath a gate, and the gate-mechanism connected with the 7.5-h.p. motor by throwing a lever; this breaks the current connection with the travel motor and makes it impossible for the car to move while it is connected with the bin gear. When the gate has delivered its proper amount of ore, it is closed, the connection is broken, and the travel motor started. In this manner as many as eight components of a charge can be dropped. At Midvale the scale-car is run on to a bridge over a charge-car and its content dropped; 24 men handle in 24 hr. 1,500

[1] Editor, *Eng. Min. J.*, 1915, C, 519.
Anderson, *op. cit.*, 1916, CI, 885.
Private notes, 1916.

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tons charge; one man feeds six furnaces. In a new plant the scale-car would be the charge-car.

115. Handling Lead.—There are three ways of removing blast furnace lead from the basin of the Arents siphon-tap: dipping, tapping, and allowing to overflow. The crude lead is usually transferred to a drossing-kettle holding from 20 to 40 tons lead, where it is freed from impurities held in suspension, and sometimes poled; the cleaned lead is then sampled and cast into bars.

Formerly the lead, resulting from charges running low in lead, was ladled from the well into a string of molds; the liquid lead in the molds was freed from floating dross by skimming; the solidified bars were transferred on to a truck, and sampled.

This method has become obsolete, as there is no good control of the level of the lead in the crucible of the furnace, and the content and distribution of precious metal and dross in the bars from different dippings are likely to be uneven, causing no end of difficulties in the valuation of the precious metal in lead bullion.

Tapping the lead from the well has therefore become common practice with charges containing about 15 per cent. Pb; this keeps the level of the lead in the crucible uniform and thus lessens the danger of formation of crusts. The lead from such charges is collected in a two-wheel spherical cast-iron pot, and this trammed to the drossing-kettle. It is also gathered in a cylindrical pot, cast iron or plate iron brick-lined; the pot either has wheels and is trundled to the drossing-kettle, or it is without wheels and is handled from an overhead electric crane. With charges containing less than 15 per cent. Pb, the lead tapped into the receiver is cast into bars, as at El Paso, Texas,[1] to be trammed to the drossing-kettle.

At the Selby Lead Works[2] there was in operation a casting system in which lead was collected alternately in two tilting kettles which ran on two sets of tracks intersecting at an angle of about 130° under the spout of the lead-well. A kettle filled with lead was run to the end of the track, and the lead discharged into a set of ten flanged, cast-iron, water-cooled molds. Such a complicated apparatus is not desired near a furnace.

A continuous overflow of lead from the well is appropriate with charges containing about 40 per cent. Pb. This method is in operation at Herculaneum, Mo., and Northport, Wash. At Herculaneum[3] the daily product of 90 tons lead from charges with 40 per cent. Pb is run into molds on a casting-wheel. The rough pigs are trucked to the refinery which contains liquating reverberatory furnaces delivering into 30-ton kettles in which the lead is poled with steam, drossed, and siphoned into molds on a casting-wheel, whence the pig-lead goes to market. A similar arrangement is that of Northport, Wash. A vertical section of the casting-wheel connected with the blast furnace (Figs. 181–183,

[1] Easter, *Tr. A. I. M. E.*, 1915, LII, 716.
[2] Bennett, *Eng. Min. J.*, 1908, LXXXVI, 83, drwgs.
[3] Pulsifer, *Min. Eng. World*, 1913, XXXIX, 1054.
Editor, *Eng. Min. J.*, 1914, XCVIII, 899.

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Table 53) is shown in Fig. 231, which needs no further comment. The wheel is moved by hand as soon as the mold in front of the spout of the well is filled. The bars of lead are removed by hand and transferred to the drossing-kettle.

A drossing-kettle of 20 tons capacity is 6 ft. 4 in. in diameter and 3 ft. deep; a 30-ton kettle 8 ft. 10 in. in diameter and 3 ft. 4 in. deep; drawings of kettles are given in §247.

Before charging the kettle and while it is being heated to receive liquid lead, the bottom is sometimes covered with ashes. The ashes make the dross drier, that

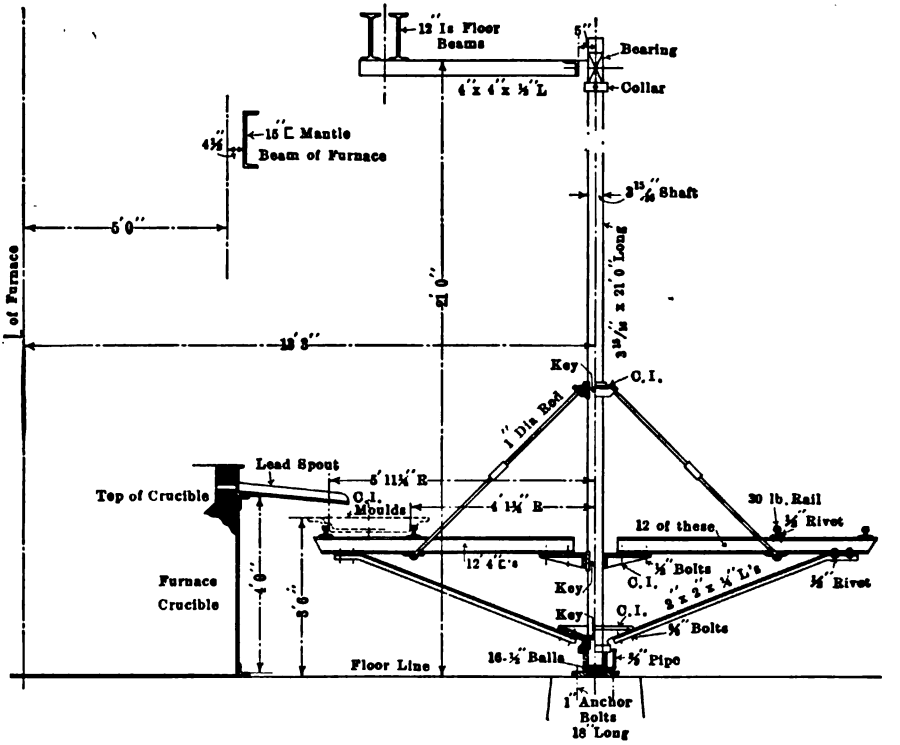

FIG. 231.—Casting-wheel for lead bullion, Northport, Wash.

is, they cause it to retain less entangled lead. When a kettle is filled with liquid lead, the temperature is allowed to fall to 480° C. before it is drossed.[1] When the dross has been removed, the lead is poled with air until the decreasing temperature reaches 360° C.; the rising dross is removed from time to time. The temperature is now raised to 380° C., when "gum-drop" samples (ladle-samples weighing about 0.5 assay-ton) are taken; it is raised further until it reaches 425° C., whereupon the lead is cast.

The dross is usually skimmed into a Howard press (§248) to remove as much as possible adhering lead. The pressed dross usually goes direct to the blast furnace charge; sometimes it is further freed from lead by liquating in a rever-

[1] Weeks, *Tr. A. I. M. E.*, 1915, LII, 711.

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beratory furnace. At some works the dry dross is allowed to accumulate and added to the blast furnace when sufficient has been collected to have a decided influence upon the Cu-content of the resulting matte.

Molding of lead is taken up in §253.

116. Poling Lead.—This term signifies the mechanical stirring of liquid lead by means of compressed air at a temperature of 400–500° C. in an open kettle. The object of the operation is to bring to the surface impurities held in suspension by the lead. When they rise to the surface they become slightly oxidized and remain on the surface of the heavier lead bath, whence they are removed as a more or less pasty mass, called dross, by skimming with a perforated disk, 10–12 in. in diameter, riveted to a pipe-handle, 6 ft. or more in length, having a T-handle.

The purification of lead by drossing is considered in §241.

Poling of lead bullion, which is usually shipped to a distant refinery, is of recent date. It has become common practice with smelteries, as, by removing the dross from the bullion, the dross remains at the smeltery where it belongs, and the lead freed from dross can be satisfactorily sampled for determining the content of precious metal.

Impurities, such as As and Sb, which are alloyed with the lead and removed only to a very small extent by poling, are taken care of in the refinery by the operation of softening (§241).

With lead that is high grade and sufficiently free from precious metal to go straight to market, poling at the smeltery is absolutely necessary, as all primary lead retains some intermingled impurities which have to be removed before it can be used in the metal industries.

Formerly a green pole was thrust into the lead. The gases and vapors set free stir up the lead and expose continually new surfaces to the oxidizing action of the air. Thus most of the Cu and S, and small amounts of As, Sb, Zn and Fe are slowly oxidized, and collect on the surface as a dross. It takes about 2 hr. to pole a kettle.

Later a crutch was used to keep a stick of green wood horizontally depressed in the lead. A crutch consists of a piece of flat-iron long enough to reach about 1½ ft. over the kettle upon which it is placed and weighted with a couple of bars of lead at either end. To the flat-iron are riveted two arms, say 2 ft. 6 in. long and 2 ft. apart, forked at the ends which, reaching into the lead, receive the wood; they are connected halfway down by a cross-piece of flat-iron. Usually the crutch with its billet of green wood is put in place before the kettle is filled, as this obviates the tedious work of depressing a green stick in hot lead.

In modern plants, which have available steam and compressed air, poling with wood has been replaced by introducing steam or preferably air near the bottom of a kettle through a 1- or 1.5-in. pipe, bent to the form of the kettle and passing downward through the lead. The pipe bent over the rim of the kettle is weighted with bars of lead, or the rim has a clamp which holds it in place. In using steam, it is important that an automatic trap be placed near the kettle to remove condensed water.

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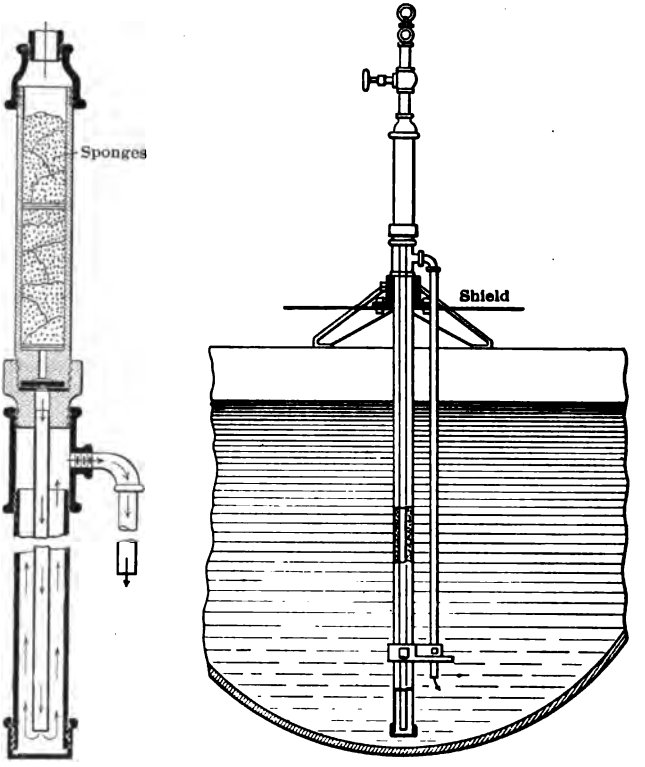
At Herculaneum, Mo.,[1] the lead is steam-poled by means of the Bardill apparatus given in Figs. 232 and 233. It shows two connecting concentric pipes. The inner pipe, open at the ends, is screwed at the top into a cap which closes the outer pipe and has an opening 0.02 in. in diameter leading into a cylindrical receiver filled with sponges and connected with pressure-water. The outer pipe forms a generating chamber from which steam is withdrawn at the side through



FIGS. 232 and 233.—Bardill poling apparatus.

an elbow and passed downward through a delivery-pipe. The apparatus is lowered into the lead, being suspended by a chain-block from an overhead trolley, and heated by the lead. Pressure-water, freed from possible suspended matter by the sponges, enters the inner pipe through the capillary opening, is vaporized in its passage downward in the inner and upward in the outer pipe, passes near the top of the latter into the delivery-pipe, and poles the lead. The apparatus is cheap, effective, and easy to handle, but care has to be had to avoid explosions.

117. Handling Matte and Slag, General.[2]—An imaginary cross-section through the crucible of an operating furnace would show the crucible filled with

[1] Bardill, *Eng. Min. J.*, 1915, C, 969.
[2] Braden, *Tr. A. I. M. E.*, 1886, XXVI, 38.

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lead and floating on top of it speiss, matte, and slag in the given order. The lead is removed through the siphon-tap at the side (see Figs. 197 and 198), and the other products are tapped together intermittently at the front into a receiver in which they separate according to their specific gravities (7, 5, 3.5 respectively) speiss at the bottom, followed by matte, and this by slag. On top of the normal matte there is found frequently an abnormal matte-like mush[1] rich in Zn, which has not been taken up by matte or slag, and is likely to cause considerable trouble.

In smelting the usual run of ores the amount of speiss produced is too small to separate from the matte; it remains suspended in or dissolved by the matte. With arsenical ores or with refinery products containing considerable amounts of As and Sb, it is generally not feasible to volatilize these two metals to such a degree that they do not form speiss, and this undesirable intermediary product is formed, often in unexpected quantities. When this is the case, the speiss has to be separated from the matte by breaking the solidified cake. In the present discussion, speiss needs no further consideration.

118. Separation of Matte and Slag.—With small furnaces treating charges which gave a low matte-fall, matte and slag were tapped into slag-pots. These were removed to the dump by slag-wheelers, allowed to solidify, and broken with sledges when cold. With the increase of S in the charge the larger amount of matte produced was taken care of by using overflow slag-pots which retained the matte and were exchanged when the level of the matte had risen to the overflow-spout. The waste slag was hauled to the dump and handled in the old way or was poured over the edge of the dump. As the furnaces became larger and the matte-fall increased, the small overflow-pots were replaced by large movable fore-hearths, and waste-slag pots introduced in sizes which required power to take them away so that the slag might be disposed of by pouring over the dump or into a granulating apparatus. The matte was tapped at intervals from the fore-hearth into a receiver where it solidified, or into ladles when it was to be brought forward by converting. This is common practice at present. In a few instances the slag-matte mixture, tapped from the blast-furnace, is transferred to a reverberatory settling furnace from which matte and slag are removed intermittently.

Though the former small-size slag-pots are not much used in ore-smelting plants, they have been retained to a certain extent in refineries having small blast furnaces for the treatment of batches of intermediary products.

Figs. 234 and 235 give a plan and section of a hand-drawn slag-pot with curved handle, 5 ft. long; the bowl has a capacity of 1.6 cu. ft.; the axle, 8½ in. long and 1 7/16 in. in diameter, runs on roller-bearings, 3 7/16 in. long and ¾ in. in diameter; 12 such pots are sufficient for a furnace 36×60 in. at tuyère-level. As the bottom of a pot is worn out more quickly than are the sides, Terhune[2] has made it exchangeable.

An overflow-pot is represented in Fig. 236, which is larger than the regular

[1] Easter, *Tr. A. I. M. E.*, 1915, LII, 716.

[2] *Tr. A. I. M. E.*, 1886–87, XV, 92.

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catch-pot, Figs. 234 and 235. Ordinarily the slag is allowed to harden from the surface down to a depth of a few inches, the inlet and overflow being kept open. The hardened slag prevents the liquid slag below from cooling, and thus promotes a good separation of matte and slag.

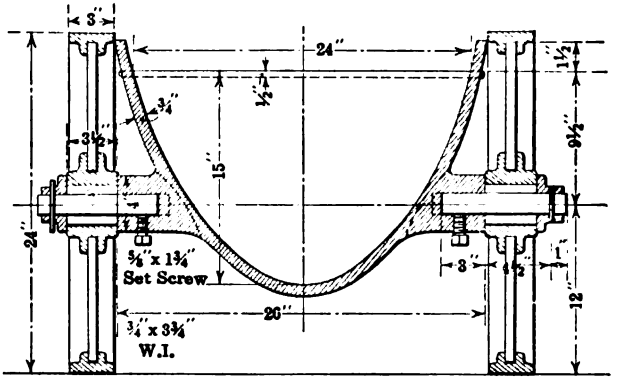

FIG. 234

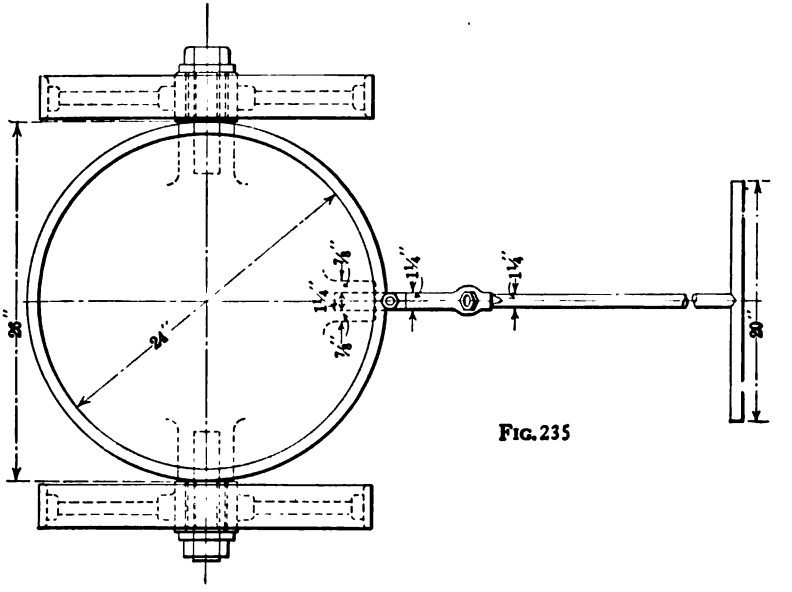

FIG. 235

FIGS. 234 and 235.—Slag-pot, hand-drawn.

In order to do away with the necessity of moving the heavy overflow-pot when filled with matte, a tap-hole[1] has been placed near the bottom as shown in Fig. 237. Matte is periodically discharged through it into an ordinary slag-pot. As this tap is likely to wear away quickly, Terhune, Fig. 238, made it detachable.

[1] Braden, *Tr. A. I. M. E.*, 1896, XVI, 38.

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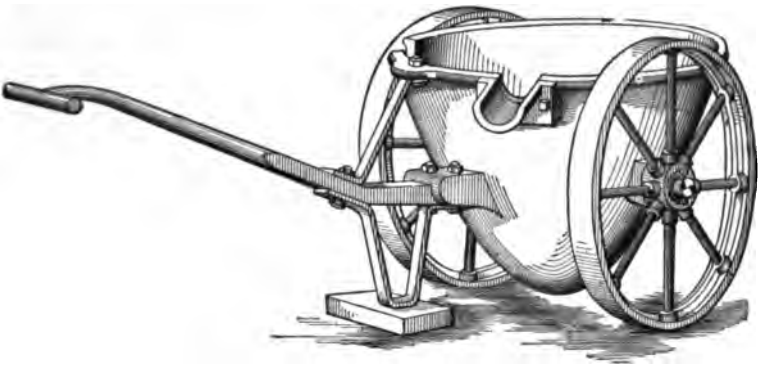

FIG. 236.—Overflow slag-pot, hand-drawn.

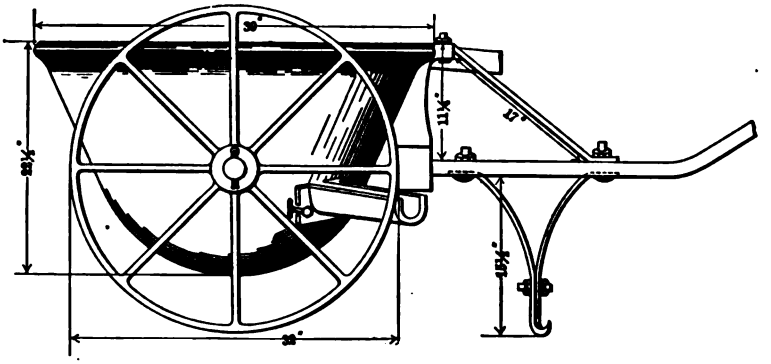

FIG. 237.—Overflow slag-pot with matte-tap, hand-drawn.

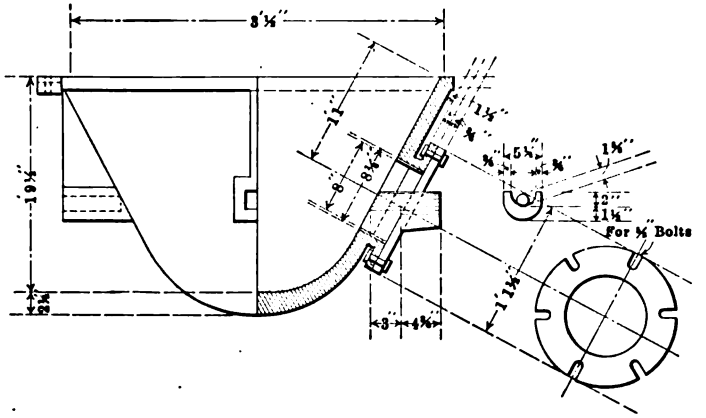

FIG. 238.—Overflow slag-pot with Terhune detachable matte-tap.

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Instead of using an overflow-pot to collect matte, there was in operation for some time a catch-pot with tap-hole above the level to which the matte was expected to reach; the still-liquid slag was tapped into a waste-slag pot, and the matte in the bottom remained undisturbed and was allowed to solidify. The slag-shells were returned to the blast furnace. Such pots went by the name of Deveureux pots, Deveureux having taken out several patents;[1] drawings are

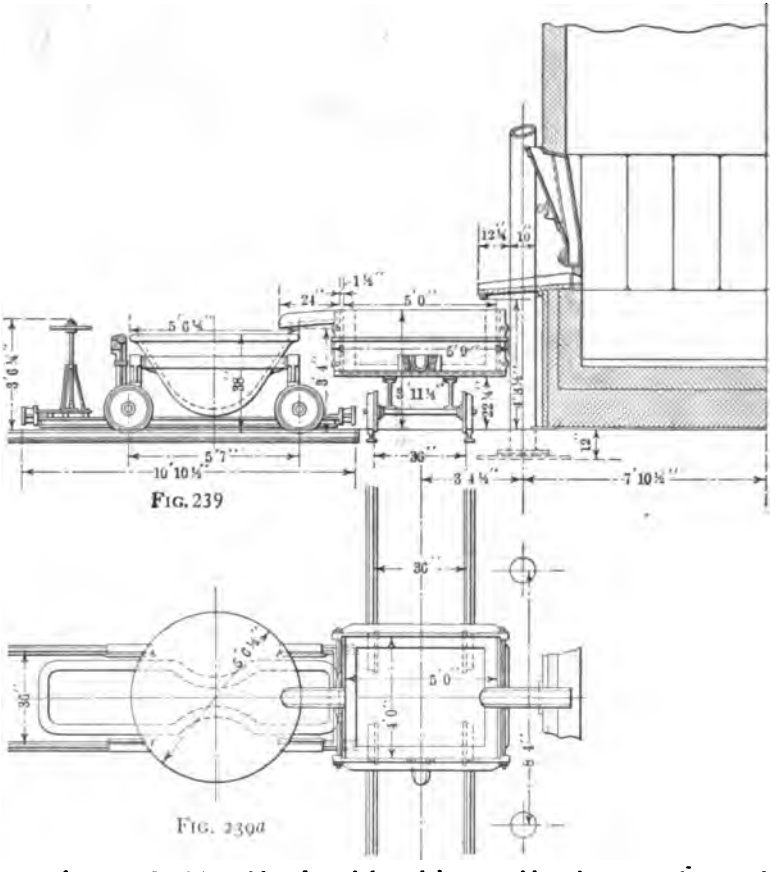

FIG. 239

FIG. 239a

FIGS. 239 and 239a.—Position of fore-hearth hearth between blast furnace and waste-slag pot.

given in the preceding edition. The principle involved is, however, of an earlier origin than is generally supposed. Kerl[2] states that such a pot was already in use in 1873 at Schemnitz, Hungary; Painter[3] reports its use at Stolberg, Prussia, at the same date, and Vivian[4] as early as 1846 tapped the liquid slag from cones placed on the dump and saved the sculls, which alone were resmelted.

[1] U. S. Patents, No. 312439, Feb. 17, 1885; No. 335114, Feb. 2, 1886.

[2] "Grundriss der Metallhüttenkunde," Felix, Leipsic, 1881, p. 100.

[3] "Report on the Metallurgy of Lead, Silver, Copper, and Zinc at the Vienna International Exhibition of 1873," Washington, 1875, p. 61.

[4] *Eng. and Min. J.*, 1881, XXXI, 249.

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At the Omaha and Grant Works, Denver, Colo., Livingstone put this principle into operation on a large scale.[1] The resmelting of slag-shells has been found to be unprofitable as long as the settling of matte has been well attended to.

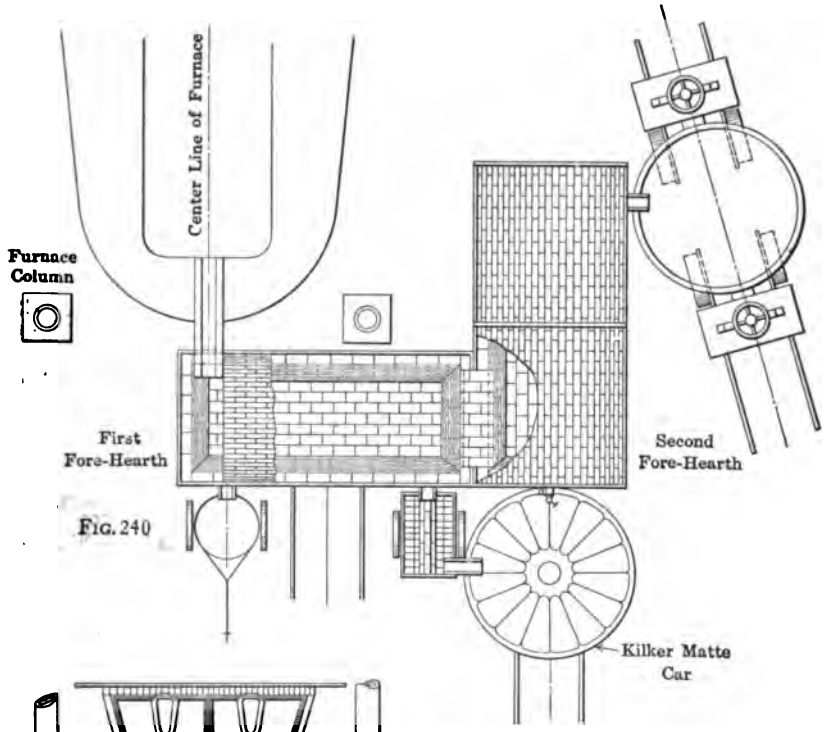

FIG. 240

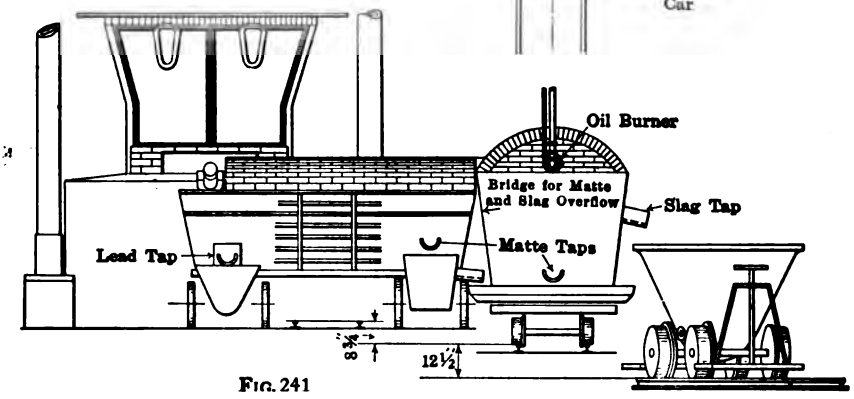

FIG. 241

FIGS. 240 and 241.—Positions of two fore-hearths in series between blast furnace and waste-slag car.

With the increase of size of furnace and of matte produced the older catch-pots have had to be replaced by larger receivers which have taken the form of a fore-hearth. The position of a single movable fore-hearth between the blast furnace and the waste-slag pot is shown in Figs. 239 and 239*a*.[2] The fore-hearth,

[1] Austin, *Eng. Min. J.*, 1895, LX, 490.

[2] Drawing of Colorado Iron Works, Denver, Colo., 1900.

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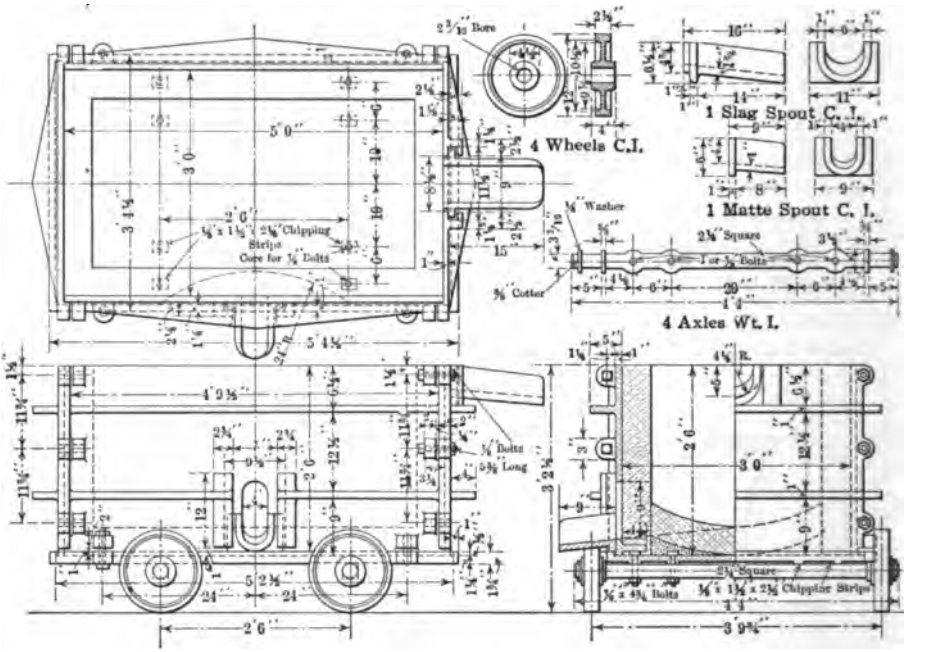
built of cast-iron plates bolted together, is intended for a small furnace; it is 4 by 5 ft. and 2 ft. deep inside the castings, and is lined with ordinary red brick. It receives the slag-matte from the blast furnace at one end and collects the matte, while the slag overflows at the other into a tilting waste-slag pot with bowl 5 ft. 6½ in. in diameter; the matte is tapped at intervals into a receiver. It will be noted that the tracks for the fore-hearth and the waste-slag pot are placed at right angles; both are 36-in. gauge.

With the practice recently inaugurated of allowing slag-shells to go to the dump instead of resmelting them in the blast furnace to collect their values, more attention has been given to perfect settling than used to be customary;



FIGS. 242 to 251.—Fore-hearth, Colorado Iron Works, Denver, Colo.

hence the tendency at present is to have two brick-lined fore-hearths in series instead of one. Such an arrangement is shown in Figs. 240 and 241. The slag-matte is tapped into the first brick-lined fore-hearth which is 9⅛ by 4¼ ft. and 2½ ft. deep outside; it has an arched roof, a lead-tap on the brick floor, and a matte-tap 7½ in. higher up. The entrained lead is tapped into a cast-iron slag-pot, and the matte into a cast-iron brick-lined receiver which overflows into a Kilker matte-car (§119). The slag-matte, which has dropped most of the matte in the first fore-hearth or settler, flows over a bridge of fire-brick into the second lined settler which is 10 ft. long, 4¾ ft. wide at the top and 3¾ ft. at the bottom, and 3½ ft. deep outside. This also has an arched roof, a matte-tap at one end, and a slag-overflow spout at the other which delivers into a waste-slag pot, 4¾ ft. in diameter. Above the matte-tap is placed an oil-burner to correct any

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cooling of the slag. In the drawing the two settlers are placed at right angles to one another; ordinarily one follows the other in a straight line.

A detailed drawing of a fore-hearth constructed by the Colorado Iron Works, Denver, Colo., is shown in Figs. 242–251. The hearth, 5 ft. 4½ in. long, 3 ft. 4½ in. wide, 2 ft. 4¼ in. deep outside, has a 1¾-in. bottom-casting, 1-in. side- and end-castings strengthened by 1-in. ribs; the bottom has up-turned 1-in. borders to receive the side- and end-plates; at the corners bottom-, side-, and end-plates are fastened together by means of ⅞-in. bolts. The hearth has two pairs of wheels with 2-ft. 6-in. centers running on a 3-ft. 9¾-in. gauge track; the slag-overflow is at one end, and the matte-tap in the middle of one side. The brick lining is shown to be 9 in. thick. Such a heavy lining is appropriate when making matte and slag which do not incrust the hearth. With the zincky ores smelted at present under conditions which require, for economic reasons, slags running high in SiO_2 and CaO, the life of a fore-hearth is too short to warrant putting in a 9-in. lining. In some instances a fore-hearth is exchanged every 24 hr., in others once every 2 or 3 days; the latter is the more common figure. With two fore-hearths in series, the first will be freed from lead and matte through two tap-holes; the second does not contain any lead. The bottom-lining of the first hearth becomes very tough on account of the lead that is present; it may require drilling and blasting. The bottom of the second fore-hearth is easily broken by dumping; the larger pieces are then passed through a crusher.

Before lining a fore-hearth, a ladle of waste slag is poured in to close the joints between the bottom- and the end- and side-plates; on top of the slag is laid a flat course of red brick (bats) in ordinary mortar; the sides also receive only a flat course. The space surrounding the matte-tap is bricked up. It ought to be stated that in ore-smelting it is not absolutely necessary to line the sides at all, but it is safer. In matte-smelting a lining is indispensable.

A re-lined fore-hearth is dried and warmed by burning in it cord wood for about 24 hr. When it has been put in place, and matte is run in, cord wood is kept burning on the rising slag, and the hearth is kept loosely covered with pieces of sheet iron; on the discharge-spout wood is kept burning to counteract the cooling effect of the air.

A fore-hearth is tapped clean before it is exchanged.

A fore-hearth with continuous slag- and matte-discharges was in operation at Nelson, B. C.,[1] but has not found use elsewhere.

In a Reverberatory Furnace the separation of matte and slag was first carried out by Rhodes in Leadville, Colo.,[2] and later by Iles in Denver, Colo.[3] It was in use at El Paso, Texas,[4] and is in operation at Mapimi, Mexico[5] and perhaps other places.

[1] Harris, *Eng. Min. J.*, 1906, LXXXI, 178; *Can. Min. Rev.*, 1906, XXVI, 141.
Hedley, *J. Can. Min. Inst.*, 1906, IX, 58.
[2] Kloz, *Eng. Min. J.*, 1897, LXIII, 358.
[3] Iles, *op. cit.*, 1899, LXVIII, 695.
[4] Vail, *op. cit.*, 1914, XCVIII, 465.
Easter, *Tr. A. I. M. E.*, 1915, LII, 716.
[5] Rice, *Eng. Min. J.*, 1908, LXXXVI, 373.

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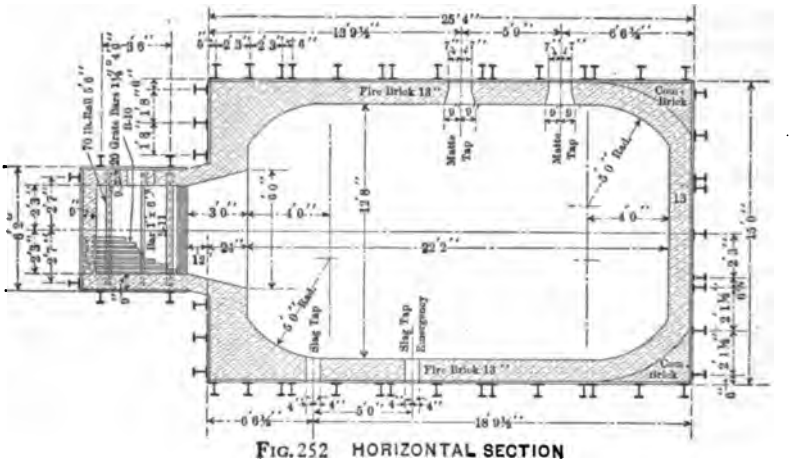

FIG. 252 HORIZONTAL SECTION

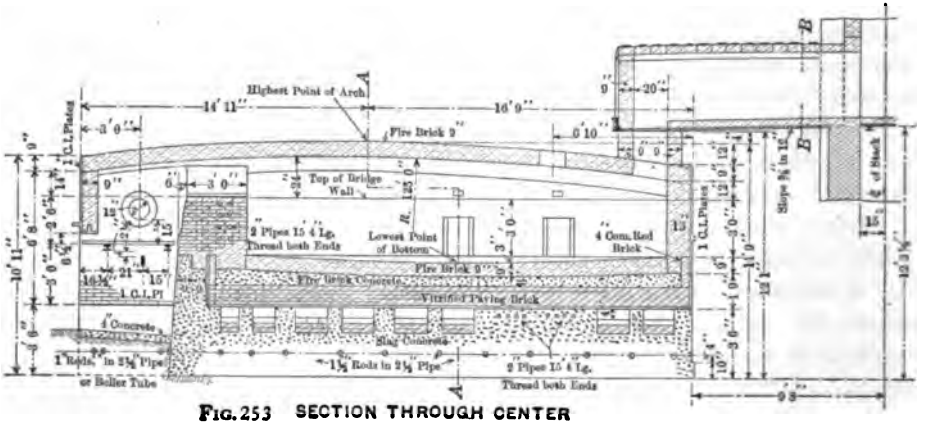
Highest Point of Arch
Fire Brick 9"
Top of Bridge Wall
2 Pipes 15 4 Lg. Thread both Ends
Lowest Point of Bottom
4 Com. Red Brick
Fire Brick 9"
Vitrified Paving Brick
Slag Concrete
1½ Rods in 2½ Pipe
2 Pipes 15 4 Lg. Thread both Ends
1 C.I. Plates
4 Concrete
1 Rods in 2½ Pipe or Boiler Tube
Slope ¾ in 12
Face of Stack

FIG. 253 SECTION THROUGH CENTER

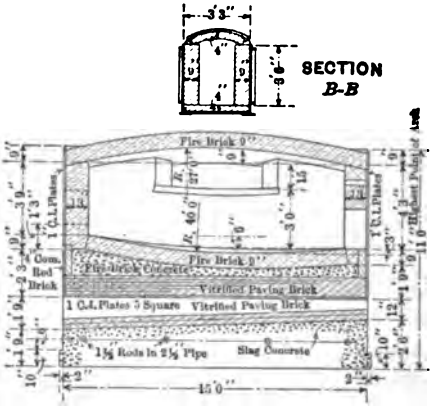

FIG. 254 SECTION *A-A*

FIGS. 252 to 254.—Reverberatory matte-separator.

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In Figs. 252–254 are given a horizontal and two vertical sections of a modern reverberatory settling furnace. The hearth, 22 ft. 2 in. by 12 ft. 8 in. and 3 ft. deep, has a capacity of about 860 cu. ft. which corresponds to about 150 tons slag. It is fired from a grate, 5 ft. 2 in. by 3 ft.; the gases, on leaving the furnace, rise in a vertical flue, placed on the center line, and then pass into a horizontal flue which ends in a chimney, 2 ft. 6 in. square. The hopper-shaped in-pour, 1 ft. 3 in. square at base, is placed near the gas-flue; two slag-taps with water-cooled tapping-blocks are on one side near the firebridge, and two similar matte-taps on the other side near the flue. The center of the slag-tap is about 18 in. above the brick bottom, that of the matte-tap about 15 in.; the lead-tap (not shown) is level with the bottom at the deepest point. The foundation is slag-concrete (5 parts slag 2 in. and under, 3 sand, 1 cement). On top of this are laid 1-in. cast-iron plates, followed by vitrified brick laid in cement, by fire-brick concrete (4 parts ground brick, 2 fire-clay, 1.5 cement), and by a full course of fire-brick which forms the working bottom. The roof is of fire-brick, 9 in. thick; the firebridge, 36 in. thick, is cooled by six 2-in. water-pipes placed in three rows toward the side of the hearth. The furnace is encased with 1-in. cast-iron plates braced by 7-in. 15-lb. I-beams tied top and bottom by 1.5-in. rods, the bottom-rods passing through 2.5-in. pipes.

The slag-matte arrives in pots traveling on an overhead track. From 45 to 60 min. is allowed for settling; the slag is tapped into bowl-shaped waste-slag pots of 55 cu. ft. capacity; the matte into a train of cast-iron molds, 2 ft. square and 5 in. deep. The furnace requires coal about 1 per cent. the weight of the slag treated, and the attendance of four men per shift.

The work of the furnace is satisfactory with a slag that is not too heavy and does not contain too many impurities. With zincky slag and with a charge making much speiss there is trouble ahead, as the zincky matte or mush floating between matte proper and slag will not flow, and as speiss settles on the bottom and often freezes at the tap-hole.

The experience at El Paso, Texas, with a reverberatory settler, recorded by Easter[1] is of considerable interest. In a furnace, with a hearth 14 by 25 ft. and a slag-matte bath of 3 ft. depth, the slag-tap was about 28 in. above the brick bottom, the matte-tap 15 in., and the lead-tap flush with the bottom. The furnace was at first coal-fired; later, oil-burners, pointing downward, were introduced through the fire-box, but were finally transferred to the roof of the furnace and played from each corner downward on to the charge. About 20 bbl. oil were burned per day. The furnace received the slag-matte in slag-pots filled at the blast furnace and transferred by hand to the in-pour. The work was satisfactory with clean materials, and the matte, being granulated (p. 273), was cheaply prepared for subsequent blast roasting. The slags from the blast furnace assayed Ag 0.7 oz. per ton, Pb 1.10 per cent. wet and 0.51 dry; an average analysis of the slag from the settler gave SiO_2 31.2, FeO 30.1, MnO 4.0, CaO 22.9, Al_2O_3 5.4, ZnO 2.8, Cu 0.23, Pb 0.83 wet and 0.51 dry, Ag 0.6 oz.

[1] *Tr. A. I. M. E.*, 1915, LII, 716.

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per ton. The matte produced averaged[1] Pb, 12.9; Cu, 19.6; Fe, 31.5 per cent. and Ag 125.8 oz. per ton. With changes in the character of the blast furnace charges there occurred troubles owing to speiss and zinc-mush, both requiring an increase of fuel in order to keep the settler in working condition; but even with higher temperatures it became filled with mush as quickly as would an ordinary fore-hearth. Feeding scrap iron[2] to decompose ZnS, when the liberated Zn will burn off, helps matters, but does not cure them. The only way to get rid of the mush was to allow the matte to accumulate until it had raised the mush sufficiently to permit raking it out through a port cut for this purpose. It was found that leady copper matte ran higher in Pb after leaving the settler than when it came from the blast furnace, it having taken up Pb from the bath on which it was floating. The difficulties with the settler became so great that the following method of working was adopted: A blast furnace is provided with a removable cast-iron box, 2 ft. 6 in. by 9 ft. 2 in. and 2 ft. deep, lined with 4.5-in. magnesite brick, which serves to collect lead when the well is choked; the furnace is connected with the box by a short water-coil spout. The slag-matte overflows through a similar spout into an oblong fore-hearth, 10 by 20 ft. inside. This is built into an iron pan which stands on rails running cross-wise, is lined at the sides and ends with 9 in. of magnesite brick, and has a removable roof composed of rail-bound fire-brick arches. Heat is supplied through two oil-burners on the center line of the roof, 6 in. back from the front, which require 7 bbl. oil per day; a small stack near the inlet withdraws the gases. The normal depth of the bath is 25 in.; of this 10–12 in. is taken up by matte, as it was found that such a mass of matte assisted in keeping the fore-hearth in good condition. The waste slag overflows into slag-cars; the matte is tapped intermittently into converter ladles. A composite sample of slag-shells produced in 5 days gave: Au trace, Ag 0.64 oz. per ton, Pb 1.46 and Cu 0.30 per cent.; and the daily 24-hr. sample assayed Au trace, Ag 0.30 oz. per ton, Pb 1.24 and Cu 0.24 per cent., showing that the separation of matte and slag was satisfactory. The average analyses of a month's products showed: Slag, SiO_2 33.7, FeO 27.9, MnO 1.4, CaO 22.9, ZnO 4.6, Al_2O_3 5.2, Pb 1.50, Cu 0.15 per cent., Ag 0.6 oz. per ton; matte 8 per cent. of charge, Fe 42.1, Pb 12.1, Cu 19.8, Zn 4.6, S 21.4 per cent., Au 0.04 and Ag 87.8 oz. per ton; lead bullion 15.7 per cent. of charge, Au 3.24 and Ag 247.6 oz. per ton. There were 2.2 per cent. S on the charge; the coke used was 14.2 per cent. A typical analysis of the zinc-mush sometimes floating on the matte is: Insol. 6.6, Fe 19.0, CaO 3.4, Zn 24.3, S 21.0, Pb 10.0, Cu 4.6 per cent.

At Mapimi[3] an 80-ton reverberatory settler receives the slag-matte from 5-ton ladles, one to a blast furnace. A ladle has a bail attached to the trunnions which serve as gliding planes when the ladle is lowered on an inclined track by means of a crane into position under the slag-spout. When filled with slag-matte, the ladle is pulled up the incline, transferred to a trough leading into the

[1] Vail, *Eng. Min. J.*, 1914, XCVIII, 468.
[2] Editor, *Met. Chem. Eng.*, 1907, V, 40.
[3] Rice, *Eng. Min. J.*, 1908, LXXXVI, 373.

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settler, and emptied. The advantages claimed for this arrangement are that the furnace is always accessible, that partial settling takes place in the 5-ton ladle, and that the amount of fume arising near the furnace is greatly reduced.

119. Handling of Matte.—There are in operation three methods of disposing of the liquid matte when it is tapped from a fore-hearth or a reverberatory settler: It is collected in suitable molds and allowed to solidify, in a receiver and granulated, or in a ladle and transferred to a converter.

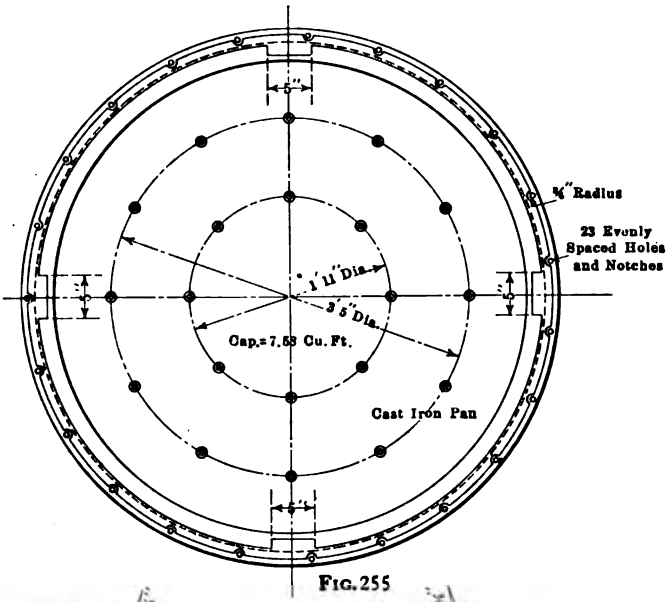

FIG. 255

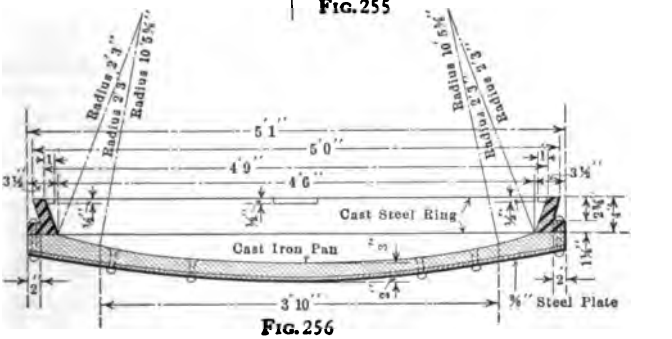

FIG. 256

FIGS. 255 and 256.—Richards matte-mold.

Formerly matte was collected in slag-pots. With small quantities of matte containing 3–5 per cent. Cu this is satisfactory, as such cakes disintegrate quickly when exposed to the air, and are easily broken. Concentrated matte, with 30 per cent. Cu and over, does not readily fall to pieces; it is tough and the breaking is a tedious piece of work. The slag-pot has therefore been replaced by flat molds. A simple form of cast-iron mold is given in Figs. 264 and 26

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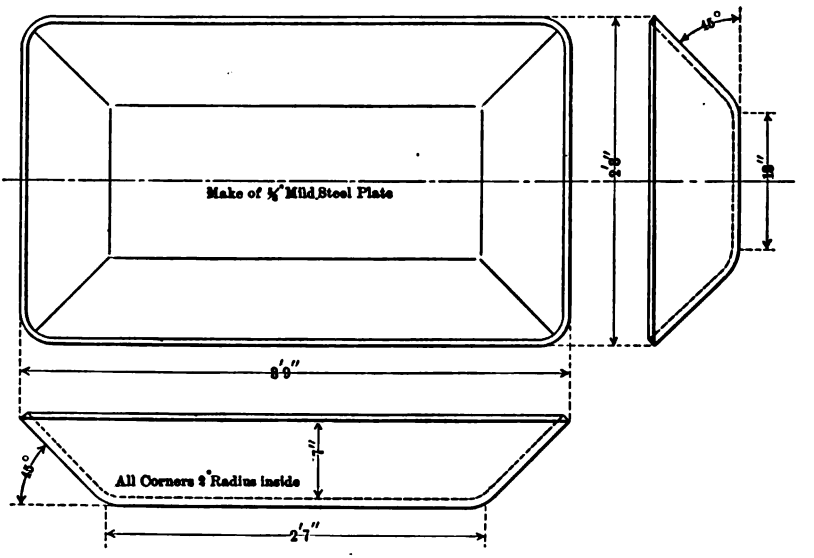
As cast iron is readily attacked by hot matte, a mold has to be clay-washed before every tap, and care must be had that all water has been expelled from the clay by the heat of the mold before this receives any matte, as otherwise explosions are likely to occur which may scatter shots of red-hot matte in all directions. Cast-iron molds are likely to be short lived on account of cracking.

The Richards mold, given in plan and section in Figs. 255 and 256, is an improved form. It consists of a steel-plate pan bolted to one of cast iron which carries a cast-steel ring. The mold has a capacity of 7.58 cu. ft. and holds therefore a little over 1 ton of matte when filled; the cake is 4 ft. 9 in. in diameter, 4 in. thick at the sides and 7 in. at the center.



FIGS. 257 to 259.—Steel matte-mold, Selby Lead Works.

In order to do away with the necessity of breaking the cake of matte by hand, Kilbowen[1] depressed into the liquid matte for a few seconds a star-pointed casting which had been clay-washed, removed it to allow the thin coat of adhering matte to solidify, and then depressed it again leaving it in position for 12–14 min. until the matte had solidified. After removing the star, the cake of matte was hoisted from the mold and transferred to a heap where, cooling quickly, it would crack along the thin walls made by the star, and furnish pieces that could be easily handled.

At the Selby Works[2] the matte mold is an oblong pan made of pressed steel; details are given in Figs. 257–259. In order to prevent the impact of a stream of matte from attacking the steel, the mold is charged with one or two shovelfuls of crushed matte before it is to be filled. Two molds are placed across the

[1] *Eng. Min. J.*, 1910, XC, 1005.
[2] Bennett, *Eng. Min. J.*, 1908, LXXXV, 252.

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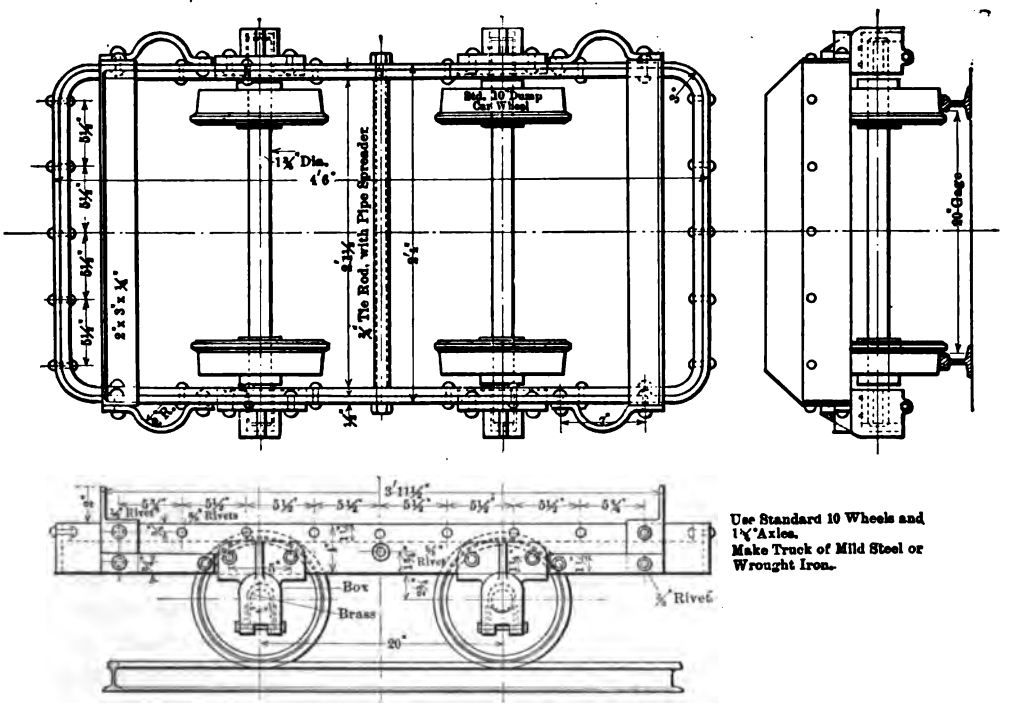
track represented in Figs. 260–262, which is built of mild steel or wrought iron. It has on each side two loops into which are inserted hooked bars 8–10 ft. long to move the car. The manipulation of the cars in front of the fore-hearth is shown in Fig. 263. There are provided two tracks, 5 ft. apart, with turn-tables at the ends and a crane at one end to handle the filled molds.

A string of connected molds has been used by Tudor[1] to prevent slopping when an empty mold takes the place of one which has been filled.


Std. 30" Dump Car Wheel
1¾" Dia.
4'6"
¾" Tie Rod, with Pipe Spreader
2'3½"
2'
2'
2'

flat tilting sectoral molds, each holding a cake of matte which is 4 in. thick and weighs about 200 lb. The weight of the car with its molds is about 6000 lb.

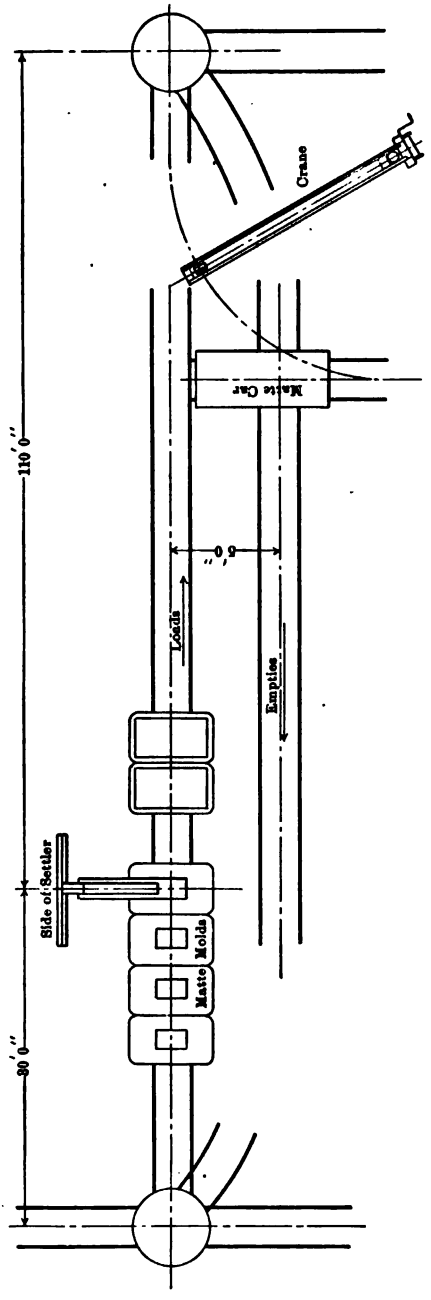


FIG. 263.—Handling of matte and matte-cars, Selby Lead Works.

In the more recent models the spider is a steel casting with 16 sectoral frames which hold the molds. In the drawing chains prevent the molds from tilting too far when they are being turned over; in the later machines a wrought-iron yoke travels around the pin, and the mold strikes against it when it is being dumped. The fore-hearth from which the matte is to be tapped has a curved spout so that the tapper can enlarge or close the tap-hole while the matte is flowing. At some works the matte flows from the fore-hearth into a small brick-lined cast-iron box on wheels, which has the curved spout. When the matte is to be tapped, the car is run underneath the curved spout, the tap-hole opened, and the spider slowly rotated with a hooked bar. The matte thus runs successively into the pans as they pass along, filling them in part during a revolution of the spider. When the tap-hole is closed, the molds will have been filled with thin layers of matte, which are easily broken. A filled car is hauled to the yard, where the matte cools sufficiently in about 30 min. to allow dumping the molds with a chisel-pointed bar. The tilted hot molds are clay-washed, returned to their horizontal positions, and the car hauled back to the fore-hearth.

The cakes of matte have to be crushed before they can be roasted or blast roasted. The cakes are broken with sledges to sizes suited for a rock breaker. This discharges into rolls, set to ½ in., which deliver the matte either directly into a trommel with ½-in. holes or first into the boot of a bucket elevator. When much

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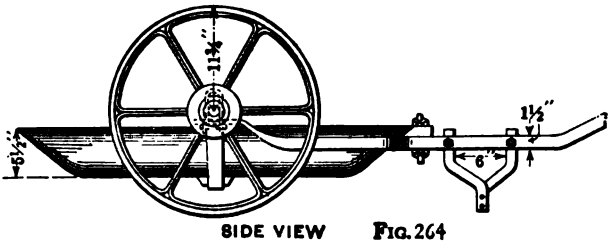
crushing has to be done, there are used frequently two pairs of rolls, for roughing and finishing, so as to expedite the work.

As this method of comminuting matte is expensive, unsanitary on account of dusting, and connected with loss from the same cause, *Granulating* has



SIDE VIEW FIG. 264

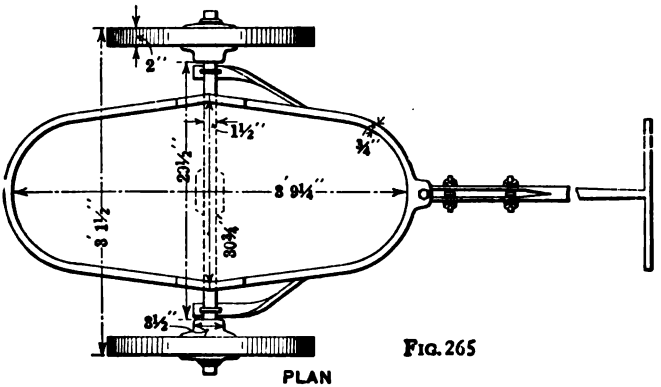


FIG. 265

PLAN

FIGS. 264 and 265.—Rhodes matte-mold.

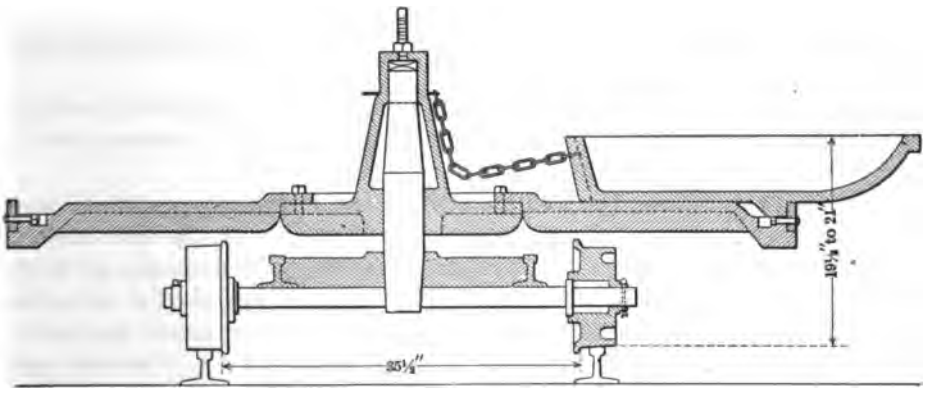


FIG. 266.—Kilker matte-car.

taken the place of crushing in a number of plants. In granulating matte, it is essential to have a thin stream, to have water running from one, or preferably two flat nozzles at a pressure of 60–100 lb., and to have sufficient settling capacity to collect the granules as well as the slime that is formed. The size of

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granules may not exceed ½ in., as otherwise they do not roast satisfactorily. It was once held that granules of lead matte were hollow and decrepitated upon heating, but this is not generally the case. Considerable care has to be had in granulating, as there is much danger from explosion, which according to Hutchinson[1] is due to the low degree of diffusivity (velocity of temperature-change), which is the ratio between thermal conductivity and specific heat. In granulating, the man who regulates the flow of matte stands behind a screen to be protected from spattering matte.

Lead matte used to be granulated at the American Smelter, Leadville, Colo.[2] The plant consisted of three wooden tanks 10 ft. long, 5 ft. wide, and 6 ft. deep filled with water up to 10 or 12 in. from the top. They were used in rotation, two being in constant use, while the third was being cleaned up. The water from the granulating tanks overflowed into a settling tank; the amount of slime caught was small. For cleaning up, each tank had a number of openings in one side, to discharge the water, and a gate in one end to remove the granulated matte. At the head of a tank, just below the top, was placed a 2-in. pipe, flattened out at the end to throw a thin, wide stream of water, the pressure of which ranged from 25 to 30 lb. per sq. in. The matte was either tapped from a settling pot or poured from an ordinary slag-pot. In the latter case iron bars were placed over the granulating tank in order to prevent the shell from falling into it and causing violent explosions. The thin stream of matte, striking the water issuing from the nozzle, was scattered into droplets and chilled before it fell into the water.

Another plant working on the same principle under a pressure of 80 lb. had two granulating tanks, 12 ft. long, 4½ ft. wide, and 2 ft. deep, discharging into one settling tank 12 ft. long, 9 ft. wide, and 2 ft. deep, which had near the discharge end a canvas partition to prevent any float-matte from being carried off.

The J. MacArthur granulating trough (§120) also does satisfactory work with matte and makes no float-matte.

At Humboldt, Ariz.,[3] copper matte is granulated with water which is delivered from a 4-in. pipe into two 3-in. pipes under a constant pressure of 60 lb. per sq. in.

At El Paso, Texas,[4] the lead matte from two reverberatory settlers, used alternately, was granulated until direct converting replaced the older method of treatment. The two settlers were placed end to end at a distance of 16 ft. and 100 ft. from the line of blast furnaces. The granulating plant of one of the settlers is shown in Figs. 267–269, and the detail of matte spouts and water-nozzles in Figs. 270–275. The matte (Pb 12.9, Cu 19.6, Fe 31.5 per cent. and

[1] *Eng. Min. J.*, 1909, LXXXVII, 1272.

[2] Bretherton, *Eng. Min. J.*, 1897, LXIII, 43.

[3] Hallett, *Colo. School Min. Mag.*, January, 1914; *Eng. Min. J.*, 1914, XCVII, 374; *Min. Science*, 1914, LXIX, 57; *Min. Sc. Press*, 1914, CVIII, 296.

[4] Vail, *Eng. Min. J.*, 1914, XCVIII, 465.
Easter, *Tr. A. I. M. E.*, 1915, LII, 722.

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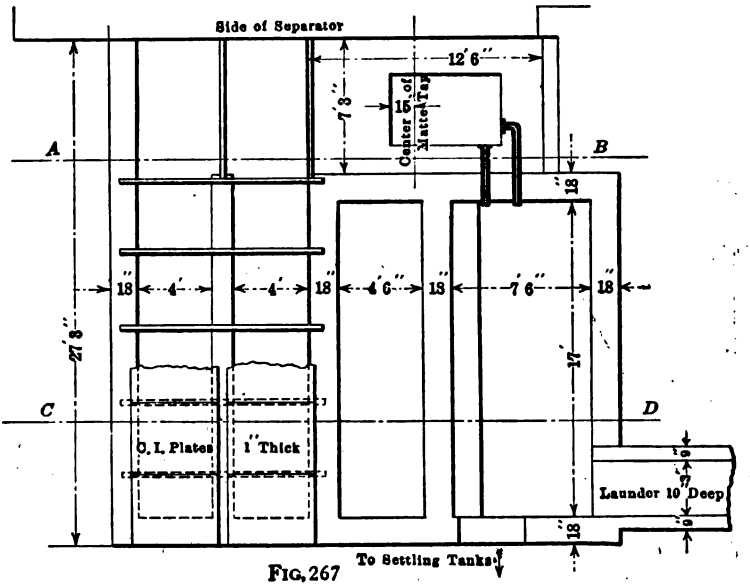


FIG. 267

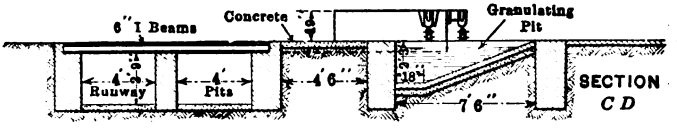


FIG. 268

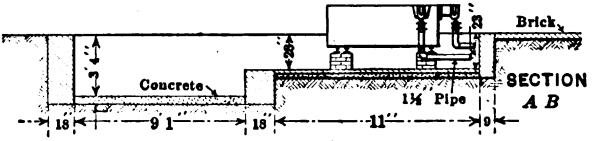
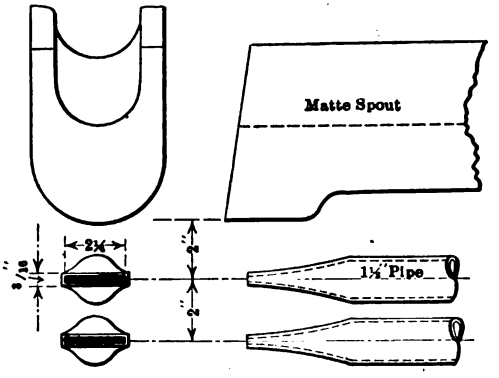


FIG. 269

FIGS. 267 to 269.—Matte granulation, El Paso, Texas.



NOZZLE DETAILS

FIGS. 270 to 275.—Details of matte-spouts and water-nozzles, El Paso, Texas.

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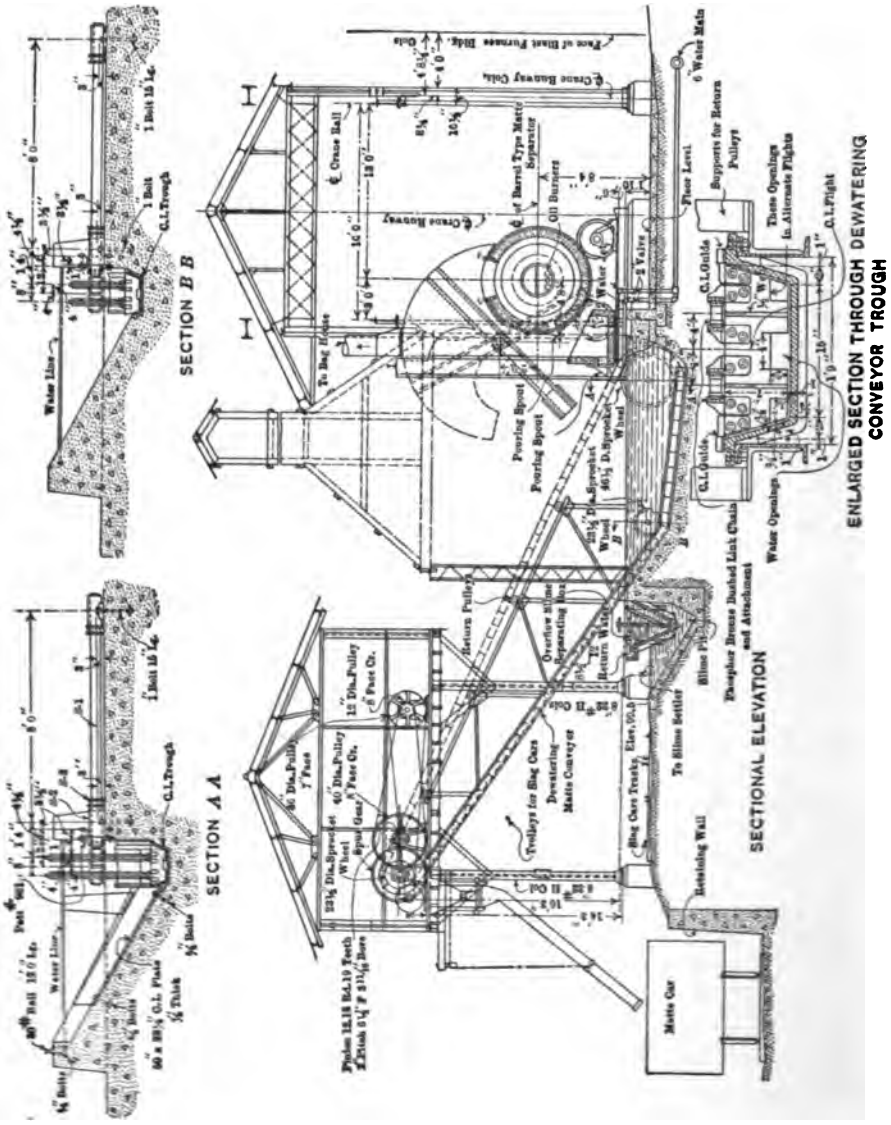
Ag 125.8 oz. per ton) was tapped through a water-cooled spout into a lined cast-iron box, 3 ft. 8 in. by 6 ft. 2 in., which served as a trap for metallic lead. The box has two cast-iron spouts which conducted the matte to the granulating pit. The two streams of matte were each cut by a double spray of water,



FIGS. 276 to 280.—Matte granulation, Herculaneum, Mo.

2¼ in. wide and 3/16 in. thick, issuing from 1½-in. pipes at a pressure of 100 lb. per sq. in. The granules were collected in a pit constantly filled with water, raked out with hoes, wheeled to railroad cars, and hauled to the Godfrey furnaces of the Huntington-Heberlein plant. The overflowing granulating water

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passed through settling tanks. In roasting, no difference was noticed in the behavior of granulated from that of crushed matte.

At Herculaneum, Mo.,[1] a plant for granulating 100 tons matte per day went into operation on March 1, 1915. The matte from four blast furnaces is collected in movable fore-hearths, 6 by 4 ft. and 22 in. deep, tapped into 6-cu. ft. iron ladles, transported by a 5-ton overhead electric crane, poured into a barrel-shaped rotating receiver of boiler iron, and granulated from this by two superposed water jets; the matte-granules are collected in a pit and transferred by means of a dewatering drag-conveyor to a car. Figs. 276–280 show the details. The receiver, 77 by 120 in. inside, is lined with 9 in. of magnesite brick, and rotated from a 50-h.p. motor. It has at each end a 20-in. opening. At one end are two oil-burners for keeping the matte fluid; they are served by air under a pressure of 30 lb. per sq. in., and consume in 24 hr. 250 gal. oil of 18–20° Bé., or 2–2.5 gal. oil per ton of granulated matte. It has been found necessary to have an oxidizing flame in spite of the fact that it has a tendency to oxidize some of the Fe, which is likely to form a floating scum interfering with the pouring. At the other end is an 18-in. steel flue connected with the bag house. A swinging hood covers the charge-opening and conducts fumes into the blast furnace flue leading into the bag house; it is swung back when a ladle of matte is to be poured into the receiver. The matte is discharged from the receiver through an opening 2 by 8 in. over a spout delivering into a pouring-box of sheet iron lined with fire-brick. The box, 18 by 36 in. and 18 in. deep inside, has at the front a slot, 2 by 8 in., delivering into a cast-iron spout which is slightly deflected in order that it may meet the horizontal jets of water at an angle of 70 or 80°. The end of the spout is 6 in. above and 12 in. in front of the upper water-nozzle. The matte builds up in the box, but leaves open a channel about 3 in. wide. The granulating water arrives under a head of 40 ft. in two 2-in. superposed pipes with nozzles $\frac{3}{8}$ by $3\frac{3}{4}$ in., placed 5 in. apart; about 100 gal. water per min. are required. The stream of matte is so regulated that the upper jet of water breaks it, and that the lower cools the granules sufficiently that on striking the main body of water they are cool enough not to cause an explosion. A stream of matte $2\frac{1}{2}$ in. in diameter, corresponding to 2.2 cu. ft. per min., has been found to work satisfactorily. If the stream is a little larger, the granules are too big; if a little smaller, the matte is likely to chill before it leaves the pouring-box. A hot matte furnishes finer granules of greater uniformity than one that is cool; with a good product, 75 per cent. will pass through a screen with openings $\frac{1}{16}$ in. in diameter. A screen analysis shows: on $\frac{1}{4}$ in.—50 per cent., on $\frac{3}{16}$—4, on $\frac{1}{8}$—14, on $\frac{1}{16}$—35, through $\frac{1}{16}$ in.—42 per cent. The granules are collected in a concrete tank, 17 ft. long by $7\frac{1}{2}$ ft. wide, which has a floor sloping at an angle of 30° and lined with 1-in. cast-iron plates; they are removed by a dewatering drag-conveyor, placed to one side and out of the line of the stream from the nozzles. The conveyor is driven at a speed of 30 ft. per min. by a 10-h.p. motor through tail and idler sprocket-wheels, both being under water. The conveyor runs in a sectional cast-iron trough, 15 in. wide and 4 in.

[1] Lindau-Smith, *Tr. A. I. M. E.*, 1916, LVI, 671.

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deep; drainage is assisted by notching the conveyor flights alternately in the center and at the ends to such an extent that the granules carry only about 5 per cent. water. The overflow from the granulating basin passes through a V-shaped settling-box, which collects the coarser particles of the fines, and then travels through one of two series of connecting settling tanks, the last being provided with a filter of excelsior. Some lead settles in the receiver, is poured into a ladle, and molded. Any lead going over with the matte is quickly noticed in that it sputters and dances on the surface of the water.

Granulation requires four men per shift for operating the receiver and the crane; it costs $0.75 per ton. The former method of comminution was, to tap the matte from the fore-hearth into slag-pots, to dump these after cooling, to break the cones of matte with sledges, and to transport the hand-broken material to the crushing division, where it was reduced to ¼-in. size. Hand-breaking and transporting cost $0.87 a ton, crushing and screening $0.56, making a total of $1.43; granulation therefore effects a saving of $0.68 per ton.

Converting matte is discussed in §185.

120. Handling of Slag.—Two methods are followed in disposing of the large amounts of slag made in a modern blast furnace; one is to have large receiving pots which are poured, the other is to granulate the slag either direct from the fore-hearth, or preferably from a receiving pot. Occasionally cakes of slag are allowed to cool on the dump, broken, and crushed to a size suitable for railroad ballast. Granulated slag frequently serves for the same purpose. Casting machines[1] are suitable, if the slag has to be resmelted, as they furnish a product which is easily handled and well suited for furnace charges.

I. *Slag Cars.*—Descriptions of a few typical waste-slag pots built by the Colorado Iron Works of Denver, Colo.; may serve as examples.

1. *Spherical Self-dumping Pot.*—The principal parts of this pot, shown in Figs. 281–283, are, a car, consisting of a steel frame with 12-in. wheels on a 24-in. track, and a central braking-mechanism; two heavy cast-iron runways curved at the ends, which form the paths on which the segmental castings at the ends of the pot travel; a cast-iron self-dumping spherical pot, 41 in. in diameter, of 9 cu. ft. capacity (=1 ton of slag), pivoted in such a manner that it will stand upright when empty, and tilt over when filled with slag; an upright post with movable cross-piece which fits into a recess near the rim of the pot and holds it in its upright position. Self-dumping pots are simple in construction, cheap, and work satisfactorily, *e.g.*, at Monterey, Mexico, when everything is in good order. If, however, through an accident the cross-piece holding the full pot in position becomes displaced, a serious accident may happen. This is the reason that most pots have a tilting mechanism which completely controls the dumping.

2. *Oval Slag-pot with Worm-gear.*—The principal parts of this pot, Figs. 284–286, are similar to those of the preceding one. The frame is built of steel rails;

[1] Braden, *Tr. A. I. M. E.*, 1896, XXVI, 51.
Austin, *Min. Sc. Press*, 1907, XCIV, 282.
Hofman, "Metallurgy of Copper," 1918, 316, 317.

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the wheels are 17 in. in diameter, run on a track with a 36-in. gauge, and are provided with a brake mechanism. The frame carries at one end a cast-iron runway for a grooved trunnion wheel, at the other a rack which meshes with the lower sectoral pinion wheel attached to the trunnion, while the upper, having smaller cogs, forms part of the worm-gear which serves to tilt the bowl by rotating the shaft with a handle. The oval bowl, 5 ft. 6½ in. in diameter and 2 ft. 7¾ in. deep with a capacity of 25 cu. ft. holding close on to 3 tons of slag, is

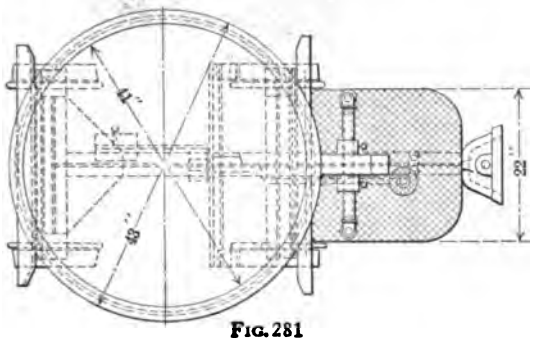

FIG. 281

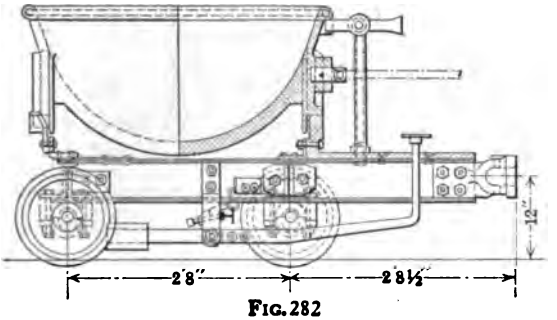

FIG. 282

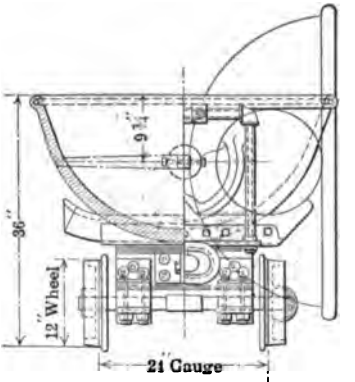

FIG. 283

FIGS. 281 to 283.—Spherical self-dumping slag-car.

carried by and attached to a heavy steel band which is connected with the trunnions. The manner of tilting and righting the bowl is clearly shown in the drawings.

3. *Elliptical Slag-pot with Releasable Worm-gear.*—The car, shown in Figs. 287–289, carries an elliptical bowl, 7 ft. 2½ in. by 5 ft. 8½ in. and 3 ft. 6 in. deep with a capacity of 45 cu. ft. or 5 tons of slag. The main features are similar to those of Figs. 284–286; the principal differences are, that a single pinion engages with the rack and meshes with the worm-shaft 35 in. in length, and that the worm-gear can be released, if this is desired toward the end of a pour to hasten the operation.

4. *Double-bowl Car.*—The car, Figs. 290–292, has two large tilting-pots, *A A*, each having a capacity of 7.38 cu. ft., and holds about 1,280 lb. of slag. The car consists of a truck-frame, *a*, with platform, *b*, brake, *c*, and railing, *d*, by which the driver holds on. The frame carries the boxes, *e*, for axles, *f*, of

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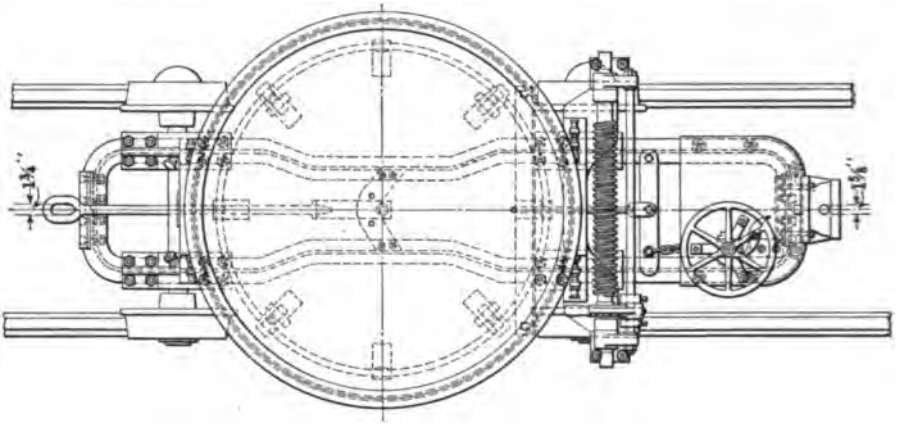

FIG. 284

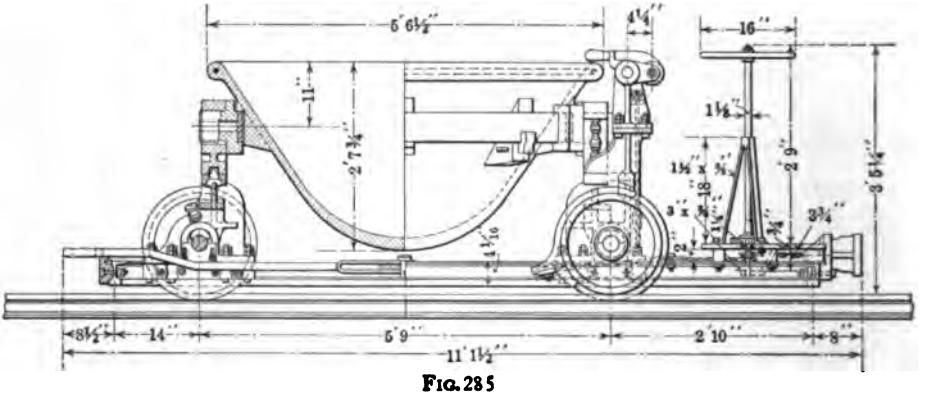

FIG. 285

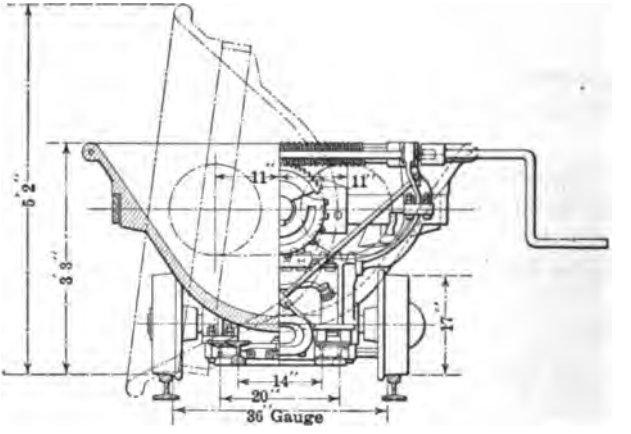

FIG. 286

FIGS. 284 to 286.—Oval slag-car with worm-gear.

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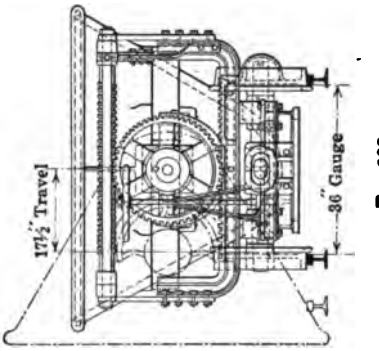

FIG. 289

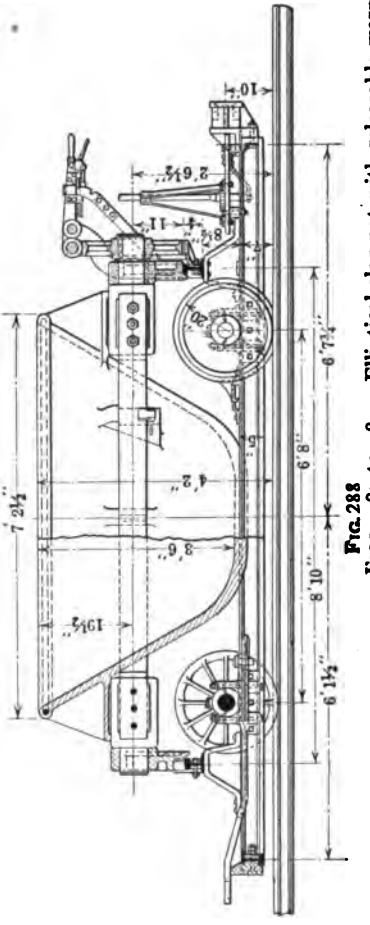

FIG. 288

FIGS. 287 to 289.—Elliptical slag-pot with releasable worm-gear.

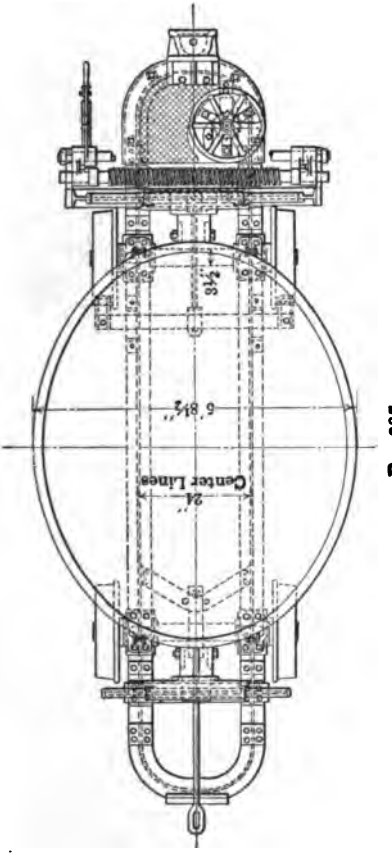

FIG. 287

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wheels, *g*. Two bridge-beams, *h*, lying transversely across the frame, *a*, serve as support for frame, *h'*, which carries central pin-socket, *m*, in which pin, *n*, is made fast by nut, *n'*. The swinging frame consists of channel-irons, *i i* (held

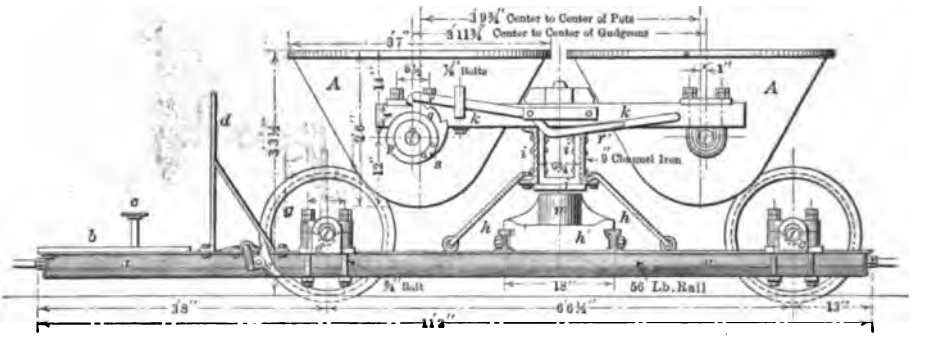


FIG. 290.—Side elevation.

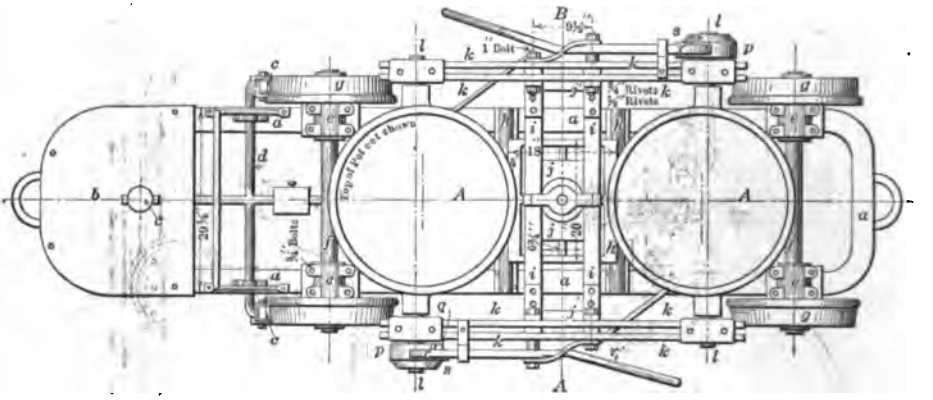
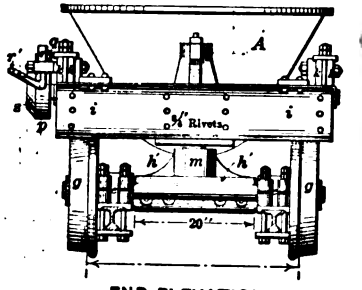
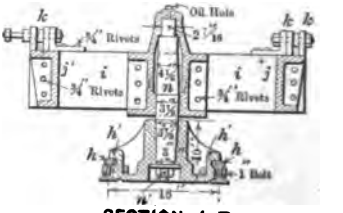


FIG. 290*a*.—Plan.



END ELEVATION
FRONT-POT REMOVED

FIG. 291



SECTION *A-B*
REVOLVED TO SIDE

FIG. 292

FIGS. 291 and 292.—Double-bowl car.

apart by central blocks, *j j*, in which are the swivel-eye and end-blocks, *j' j'*) and beams, *k k* (resting on the channel-irons); the latter have on their under side the bearings of trunnions, *l l*, of the tilting-pots, *A A*. These are pivoted

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out of center, so that when in their normal position they may lean against stop-pieces, *o*. They are held in position by pawls, *r*, with disengaging handles, *r'*, which fasten into teeth, *q*, of the projecting head or collar, *p*, of the trunnions. This collar also has the holes, *s*, for operating-bar, *t*, with which the pots are tilted. The weight complete of truck and pots is 5000 lb. When in their normal position the pots are placed as shown in Figs. 289 and 290. When they are to be emptied, frame, *iikk*, is swung 90° on the swivel, the pots are emptied together on both sides of the track; or, if the slag is to be discharged only on one side, the frame is returned to its normal place after one pot has been tilted and the other swung into position.

5. *Illustrations of other waste-slag cars* have been published by Henrich,[1] Bennetts and Jones,[2] Colorado Iron Works,[3] Power and Mining Machinery Co.,[4] Shelby.[5]

II. *Slag-granulation.*—This method of disposing of waste slag is not very common with lead smelteries. The principles involved are the same as those obtaining in matte-granulation. In carrying out the operation, there is required a larger amount of water on account of the greater amounts of material to be treated; fortunately there is less danger of explosion.

The slag can be granulated as it flows from the spout of the fore-hearth; it is more frequently collected in a waste-slag pot and granulated from this. The former method is not uncommon with copper smelteries, but is rarely found in a lead plant,[6] as it is desirable to have the furnace floor as free as possible from incumbrances, and as usually there is not enough fall available to have the granulating plant on a lower level. In the few instances in which granulating has been or is being carried on, the slag is hauled to a granulating division.

The granulating plant of the former Hanauer Works, Salt Lake City, Utah,[7] is shown in Figs. 293 and 294. Its essential features are a wooden trough, *a*, 2 ft. wide and 6 in. deep, inside measurements, the bottom of which is covered with ¾-in. cast-iron plates. A stream of water of considerable velocity running through it receives the slag, poured in from the top in a stream 1¼ in. in diameter, granulates it, and carries it off into a water-tight pit, *b*, from which a rubber bucket elevator, *c*, raises it 40 ft. into a storage bin large enough to hold the slag produced in 12 hr., say 100 tons, thus doing away with night-work in loading into the railroad cars, which carry away the slag to be used as ballast. The trough is placed underground at a pitch of ¼ in. to the foot and 10 ft. in front of the furnace building and parallel with it. The depth of the chute underground where it receives the slag is somewhat less than 3 ft., as if deeper the slag solidifies on the bottom. Near the face of the dump are chutes, covered by flanged cast-iron plates, one to each furnace, for discharging the slag into

[1] *Tr. A. I. M. E.*, 1895, XXV, 97, Ducktown, Tenn.

[2] *Op. cit.*, 1906, XXXVI, 223; *Eng. Min. J.*, 1906, LXXXII, 505, Tacoma, Wash.

[3] Mining Reporter, 1906, LIV, 168.

[4] *Eng. Min. J.*, 1906, LXXXI, 847; *Iron Age*, 1906, LXXVII, 1389, Garfield, Utah.

[5] *Eng. Min. J.*, 1909, LXXXVII, 204, Cananea, Mexico.

[6] Illustrations of such plant as at Laurium, Greece, are given in Collins' "Lead," 228.

[7] Terhune, *School Min. Quart.*, 1893–94, XV, 108, supplemented by private notes.

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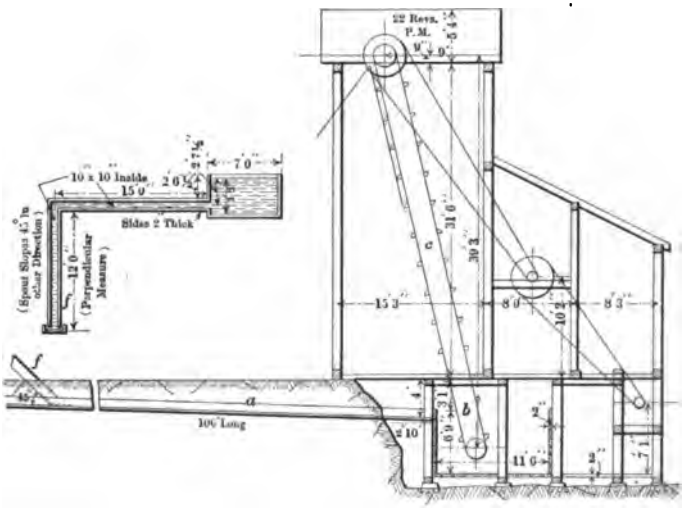
the trough. The water, which arrives in flume, *d*, is discharged at the side through closed horizontal box, *e*, into inclined box, *f*, which is open at the top. The amount of water required is 2,400 gal. per min. Very little steam is generated and hardly any noise is made, provided no matte is carried off with the slag, as matte coming in contact with water causes explosions. The cast-iron plates in the trough are worn thin in about 6 months. The elevator is driven by a 4-h.p. Leffel mining wheel. The belt, traveling at a speed of 250 ft. per min., is 8 in. wide and 8-ply, and lasts about a year; the buckets are of malleable castings and have ⅜-in. holes in the ends for the water to drain off. The slag granules are small enough to pass a ¼-in. sieve; whatever parts they strike will be quickly worn out, if not protected by iron plates.



FIGS. 293 and 294.—Terhune slag-granulating plant, Hanauer Works, Salt Lake City, Utah.

The MacArthur granulating trough, formerly used by the Canadian Copper Co., Sudbury, Ont., is shown in Figs. 295–301. The trough, *a*; is mounted on wheels so that it can be easily pushed to one side, when the movable fore-hearth, *b*, receiving the slag-matte mixture from the blast furnace, *c*, has to be exchanged, an operation taking about 10 min. The trough consists of two parts, the granulator proper, *d*, and the trough, *e*, which conveys the granulated slag to the mouth of an underground flume (not shown) 50 ft. long, which discharges into a drainage-pit whence the slag is raised by means of a bucket elevator and emptied into railroad cars. The granulator, *d*, consists of a trough-shaped casting, *f*, with side-flanges inclosed by a cast-iron jacket, *g*, of similar shape so as to leave a water-space, *h*, between them. To the upper end of the jacket is bolted the back plate with flanged spout, *i*. Through the flanges pass bolts, *j*, the joints being made water-tight with rubber gaskets. The granulating water, which is the blast furnace water-jacket overflow, and has a temperature of about 65° C., enters the water-space at *k*, rises through the opening, *l*, and being deflected by

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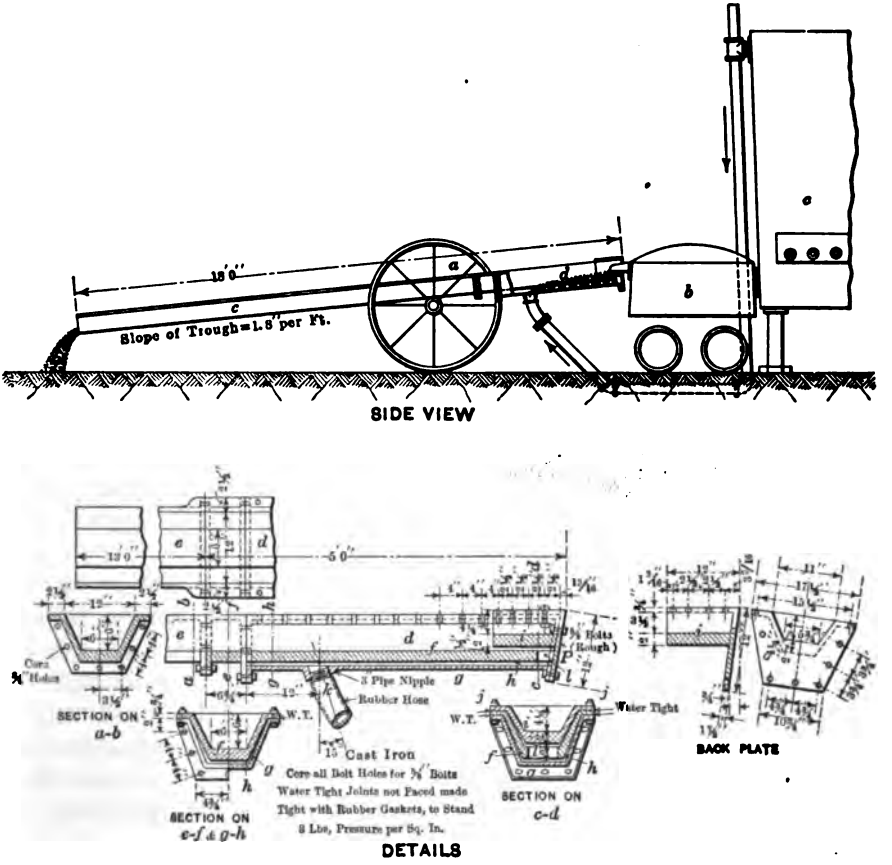
the spout, *i*, passes down under it in a thin stream. The slag flows in a similarly thin stream over the spout, *i*, and slag and water come into quiet and smooth contact, both flowing in the same direction. There is no steam, splash, or noise, and no danger of explosion. As the slag is not dropped into the water, it does not adhere to the trough-bottom and form a semi-liquid ball sure to explode. The granulated slag is uniformly coarse. About 100 tons of slag were granulated with 60 gal. of water per min., having a head of 16 ft.; but



FIGS. 295 to 301.—McArthur granulating trough.

the capacity of the granulator is half as great again as this. The apparatus is also suited for granulating matte, if this is tapped into a receiver and poured in a thin stream on to the spout.

Ordinarily slag is granulated by having a flat jet of pressure-water impinge upon a thin stream of slag as is usually the case with matte. At the Mond Nickel Works, Victoria Mines, Ont.[1] the arrangements for granulating slag and conveying the granules to the dump, shown in Figs. 302–305, were in operation before the plant was dismantled. The slag coming from the settling fore-hearth

[1] Hixon, *Eng. Min. J.*, 1906, LXXXIII, 553.

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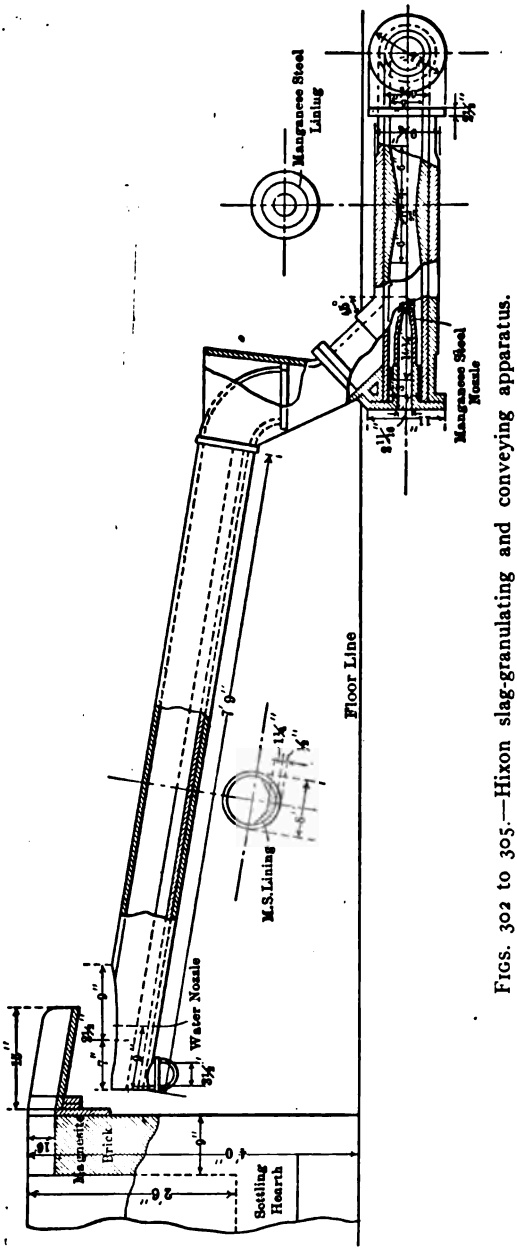
is granulated by means of water flowing in a 3-in. pipe under a head of 20 lb. per sq. in., and issuing from a nozzle, ¾ by 3⅛ in. and 9 in. long. Granulated slag and water travel for 7 ft. 9 in. through an inclined 8-in. cast-iron pipe of which the lower side has a lining of manganese steel and enter one leg of a Y-shaped casting. In the other leg, is placed a pipe $2\frac{13}{16}$ in. in diameter with manganese-steel nozzle $1\frac{15}{16}$ in. in diameter. The pipe is connected with a 5-in. main which receives from a cross-compound duplex pump 1000 gal. water per min. at a pressure of 150 lb. per sq. in. The jet of pressure-water strikes the mixture of granules and water, and drives it through the manganese-steel lining of a 5-in. cast-iron discharge-pipe which carries it to the dump. This discharge-pipe is lengthened as the dump grows. Two men on a shift attend to the apparatus which handles the slag from a blast furnace treating in 24 hr. 250 tons charge.



FIGS. 302 to 305.—Hixon slag-granulating and conveying apparatus.

Simple slag- and matte-granulating apparatus for small-size furnaces have been sketched and described by Barbour.[1]

121. Removal of Dust and Fume from Furnace Building.[2]—In a blast furnace smelting plant dust and fume arise from the handling of fine materials, from temporary openings in a furnace, and from heated and molten materials. Dust is set free mainly on the charging floor, and to some extent also a small

[1] *Eng. Min. J.*, 1915, XCIX, 239.

[2] Messiter, *Min. Sc. Press*, 1908, XCVII, 26.

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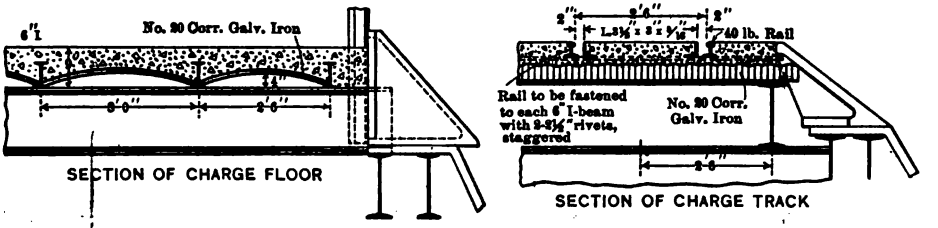
amount of fume. These have to be removed by systematic ventilation. Usually the roof of a charging floor has a ventilating monitor which may be provided with louvres. Messiter[1] states that wind blowing into a monitor strikes the under side of the roof and, swerving downward, carries with it dust and fume; he suggests that monitor openings be equipped with balanced swing-doors connected across the monitor by rods or other devices in order that when the doors on the wind-side are closed, those on the other will be open. In this way air laden with dust and fume will always leave the building through the monitor but will never enter.

For proper ventilation it is further necessary to avoid strong horizontal currents of air which are likely to blow dust and fume away from their proper channels; the air becomes mixed with the dust- and fume-carrying gases, dilutes them, and makes it necessary to handle larger volumes. It is therefore desirable to have partition walls which prevent air sweeping through from side to side.



FIGS. 306 and 307.—Blast furnace feed-floor, galvanized iron and concrete.

The feed-floor of a blast furnace is usually built of heavy planks. In time the joints become loose, especially those near the furnace, so that dust drops from the feed-floor, and fume rises from the furnace floor. A charging floor built of arches of corrugated galvanized iron covered with concrete is shown in Figs. 306 and 307.[2] It is dust- and fire-proof, and has given complete satisfaction.

The furnace-floor of the older plants used to be filled more or less with dust, fume, and smoke. At Hoboken-les-Anvers,[3] the pots collecting matte and slag are placed in a passage beneath the furnace floor. In the United States a wooden or sheet-iron hood, surrounding the front of a furnace and ending in a flue which rises through the feed-floor, delivers charged gases into the open. This is a partial remedy. Messiter[4] has called attention to the necessity of collecting smoke as soon as it is formed, before it is diluted with air. This principle is now followed at most smelteries; the large hoods serving to carry away fume which has not been caught at the source.

The first successful device for withdrawing the fume arising in tapping matte

[1] *Loc. cit.*
[2] Barbour, *Eng. Min. J.*, 1914, XCVIII, 1098.
[3] Libert and Firket, *Ann. Min. Belg.*, 1913, XVIII, 484.
[4] *Loc. cit.*

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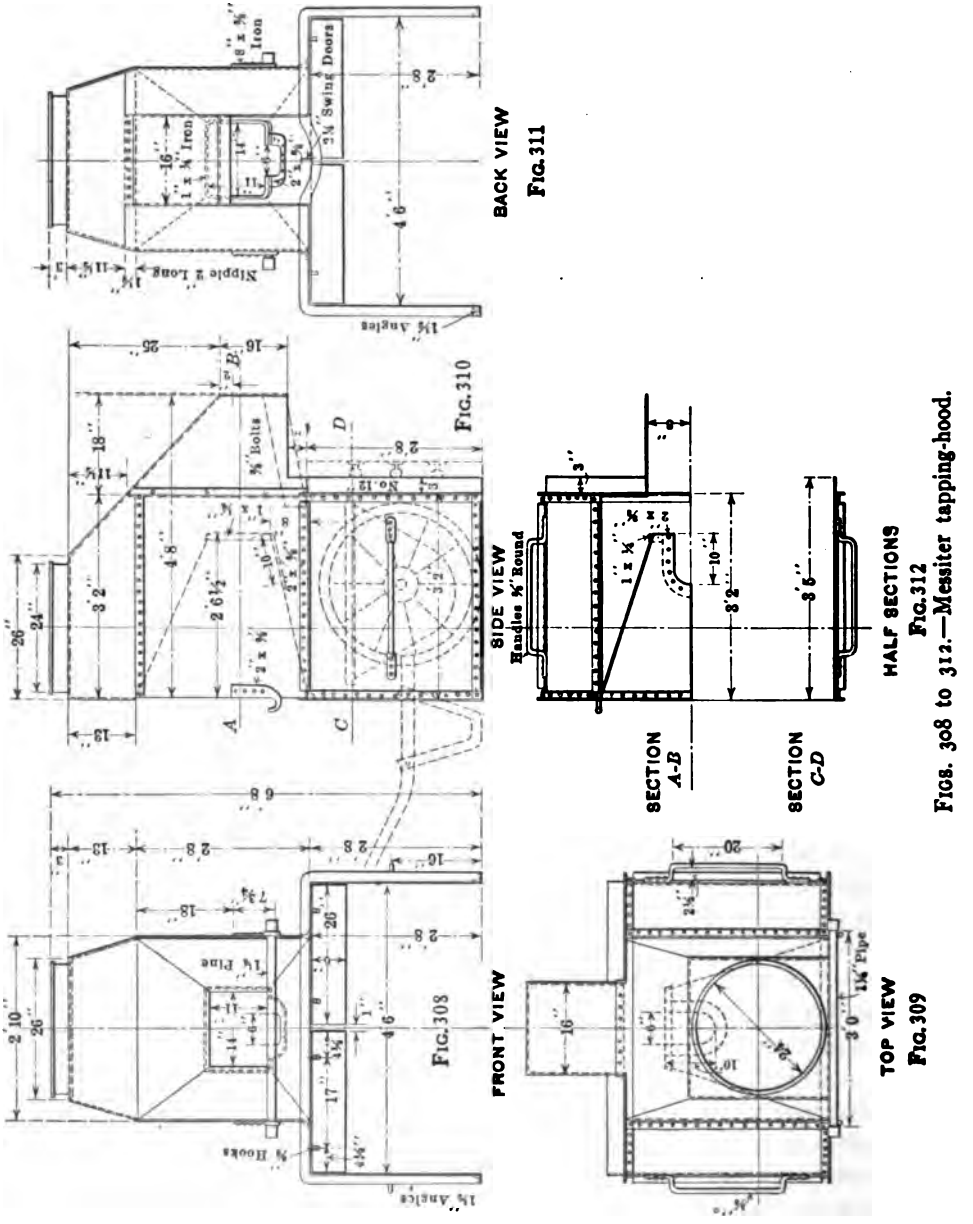
and slag is the *Tapping-Hood of Messiter*[1] which was first used at Arkansas Valley plant, Leadville, Colo., and later at El Paso, Texas, and other places. The



FIGS. 308 to 312.—Messiter tapping-hood.

hood, shown in Figs. 308–312, consists of a sheet-iron box, approximately cubical and open at top and bottom, which rests on a rectangular angle-iron support

[1] *Eng. Min. J.*, 1900, LXIX, 317.

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having the sides closed with pieces of sheet iron provided with handles. The back of the box is extended to meet the jackets surrounding the tap-hole; the

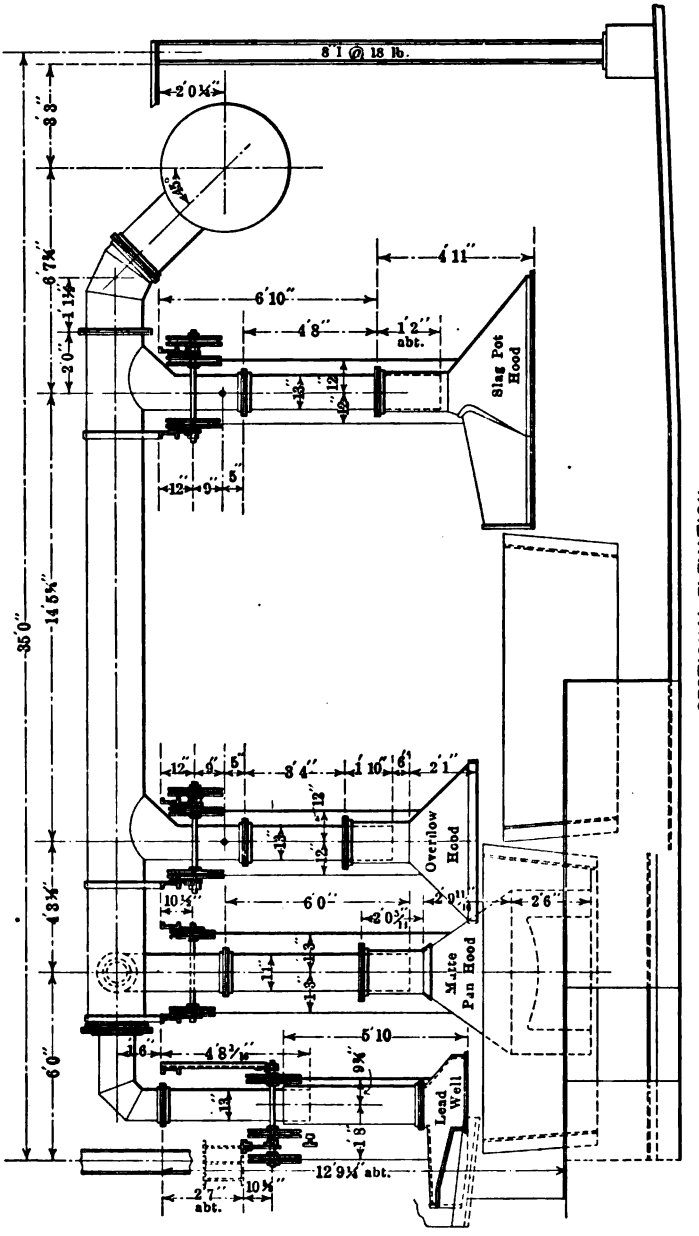

FIG. 313.—Ventilation of furnace-floor.

front has the form of a truncated pyramid with sides extending to near the back of the hood and converging toward the tap-hole. A pipe, 24 in. in diameter and 35–40 ft. long, placed on the hood, furnishes all the draft necessary to carry off

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the fumes. Across the front of the box lies a pipe to serve as a guide for the tapping-bar and plugging-rod. The angle-frame carries two swinging doors with 1-in. slot intervening which permits the tapper to follow the rising of the slag in the pot. The work of the hood is astonishingly effective.

At present similar hoods are placed everywhere around a blast furnace where fumes arise; the pipes from several hoods end in a main which is connected with an exhaust fan delivering dust and fume into a bag house. At Midvale, Utah,[1] the dust and fume collected monthly from four blast furnaces in a bag house amounted to 22 tons with an assay-value of Pb 42 and Zn 22 per cent. The collection of fume on the furnace floor is therefore not only a sanitary measure, but forms a profitable operation.

This disposition of the collecting apparatus around a furnace at the Selby Lead Works has been shown in Figs. 197 and 198.

The detail of another plant is shown in Fig. 313. The blast furnace (Figs. 240 and 241) has a lead-well at the side and two fore-hearths in series at the front. The lead-well has a hood extending over the basin and spout; the hood, balanced by counter-weights, ends in a pipe which passes telescope-fashion over the stationary branch-pipe of the exhaust main. The fumes arising from the matte-pan, when the fore-hearth is being tapped, are taken care of in a similar way; the same is the case with the overflow from one fore-hearth to the other. The hood for the waste-slag pot covers the pot as well as the spout delivering the slag. The contrast of the atmosphere in an old plant and a modern provided with proper ventilating facilities is striking to the eye; the diminution of cases of lead-poisoning is very evident, and, as shown above, the amount of fume recovered in the bag house pays for the cost.

122. Blast Furnace Plant.—So far there have been discussed the details of the roasting apparatus and operations, and the structural features of the blast furnace and its accessory apparatus. There remain to be considered the area of plant, power, light, water, etc., in one word, the engineering features, and the cost.

The only new plant which has modern methods of handling materials is that of the Bunker Hill and Sullivan Mining and Concentrating Co. at Kellogg, Idaho.[2] The details of a modern smelting plant planned by H. V. Croll of the Traylor Engineering and Manufacturing Co., Allentown, Pa., for "Blank" will give an insight into the leading features and the cost of a smeltery meeting present requirements.

1. *Bunker Hill Silver-lead Smeltery.*[3]—A plan and vertical section are given in Figs. 314 and 315; details of the Wedge roasting kiln and the Dwight-Lloyd sintering machines have been shown in Figs. 169–171, and those of the blast furnace in Figs. 184 and 185. The plant, which has a capacity of 600 tons ore per day, covers an area of 30 acres; it is built on a terraced site. The ores are sam-

[1] Private notes, June, 1916.

[2] Editor, *Eng. Min. J.*, 1916, CI, 868, plan, dimensioned.
Labarthe, *Min. Sc. Press*, 1917, CXIV, 155.

[3] Designed by Bradley, Bruff, and Labarthe, San Francisco, Cal.

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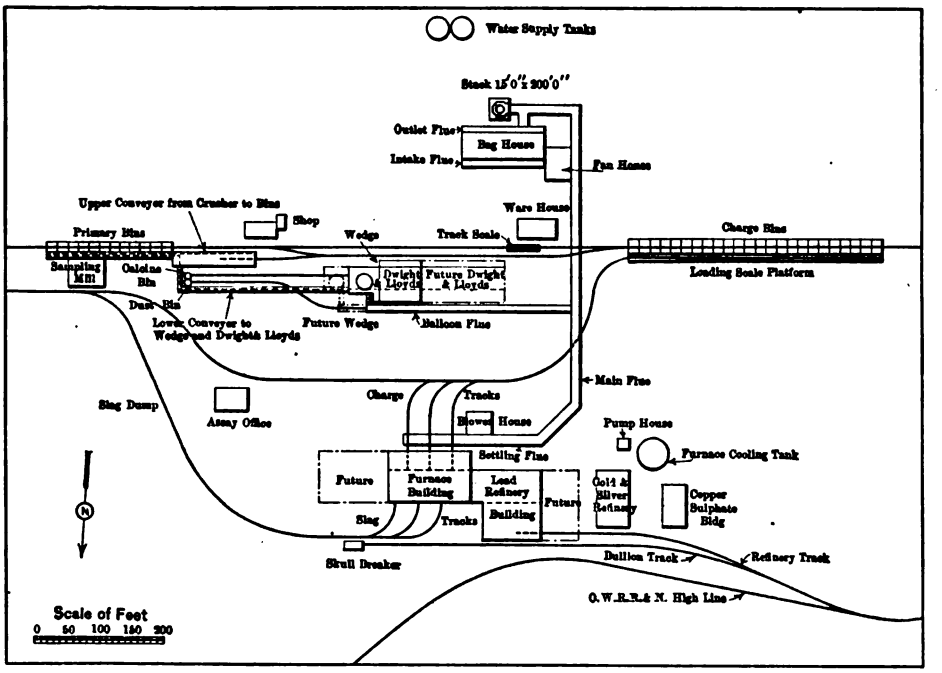
pled, roasted and blast roasted, and smelted; the lead bullion is desilverized by the Parkes process; and the doré silver parted with sulphuric acid.

In the center of the slope and at the east end, ores and concentrates, arriving in cars, are delivered into 12 receiving primary bins from which they are transferred by belt conveyor to the

Crushing and Sampling Department.—This has a capacity of 50 tons per hr.; it is equipped with a 10- by 20-in. Blake crusher, two sets of 36- by 14-in.



GENERAL PLAN

FIGS. 314 and 315.—Bunker Hill silver-lead smeltery.

rolls, one set of 12- by 12-in. rolls, a 48- by 72-in. trommel, and four Vezin samplers. The sampled ore is transferred by belt conveyors to eight storage-bins provided with apron feeders to regulate the proportions that are to be fed to a belt conveyor delivering to the storage- and mixing-bins of the single Wedge roasting-kiln and the four Dwight-Lloyd sintering machines.

Roasting Department.—The building, 112 by 57 ft., of steel construction, contains one 21 ft. 6 in. Wedge roasting kiln and four 42-in. standard Dwight-Lloyd sintering machines, shown in Figs. 169–171. The Wedge furnace rough roasts high-S ores and matte, the Dwight-Lloyd machines blast roast mixtures that are to be smelted in the blast furnaces. The product of the Wedge furnace goes into the Dwight-Lloyd charges. Both furnaces receive their charges through belt conveyors and automatic trippers in large storage-hoppers; the gases pass through an elevated balloon flue, 9 ft. 3/8 in. in diameter

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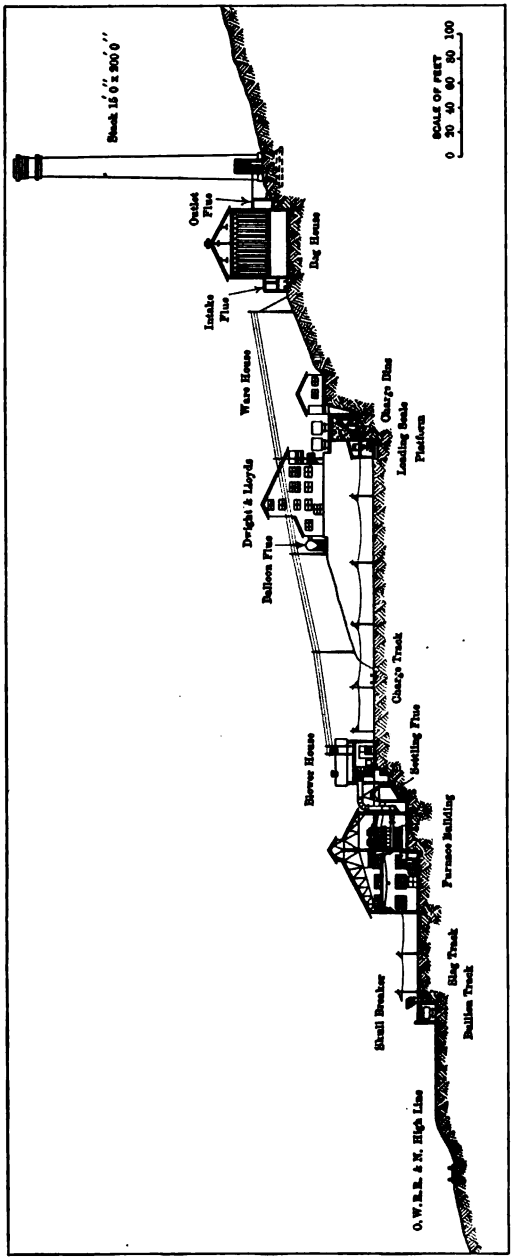
and 400 ft. long, into a Cottrell precipitator having a capacity of 60,000 cu. ft. gas per min. The product of the Dwight-Lloyd machines is collected in hopper-bottom steel cars and unloaded into concrete charge-bins. There are provided 24 charge-bins at the west end of the plant, each with a capacity of 3400 cu. ft.; of the bins, 8, built of concrete, are for roasted material; 8, built of timber, for limestone, crude ore, slag, intermediary products, etc.; and 8, also of timber, for coke. Each bin has two discharges provided with arc-gates; under each gate are hopper-scales of 4 tons capacity. The weighed charges are held in the hoppers until they are delivered to the blast furnace charge-cars. Beside the 400 tons coke held in the 8 bins, there is provided back of the charge-bins a storage place for 3000 tons coke.



SECTIONAL ELEVATION

FIGS. 314 and 315.—Bunker Hill silver-lead smeltery.

Blast Furnace Department.—The building, 80 by 216 ft., is of steel construction. It is provided with a 20-ton 4-motor traveling crane which also serves the refinery; contains three blast furnaces, 42 by 180 in., with underfloor down-takes, shown in Figs. 184 and 185. The charge-cars are hauled by 4-ton electric locomotives to the feed-floor, and bottom-discharged. The fore-hearths, of cast-steel, are handled by means of the crane; the receiving pots for the lead bullion are transferred

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by the crane to the refinery; the waste slag goes in steel collecting pots of 130 cu. ft. capacity to the dump; a skull-breaker reduces the return-slag to suitable size; the gases enter a settling-flue which delivers into a main flue, 600 ft. long, whence they pass into the fan house. This contains two electrically-driven fans, each of 60,000 cu. ft. capacity, one being held in reserve. The fan forces the gases into a bag house, 56 by 130 ft., of brick and steel construction. Beneath the thimble-floor are 12 bays; above it three chambers, each with 400 bags. Leaving the bag house, the gases are carried off by a stack 15 by 200 ft.

Power Plant.—This is housed in a brick and steel building, 35 by 40 ft. It receives an alternating current at 2200 volts. Blast is furnished by two Ingersoll-Rand turbo blowers with a capacity of 12,000 cu. ft. air per min. at a pressure of 48 oz. per sq. in.; the blowers are geared to two General Electric 200-h.p. induction motors. Two 75-kw. motor generator sets furnish the direct current for the trolley locomotives and electric cranes; one motor serves as an auxiliary unit.

General Equipment.—This consists of a brick assay-office and laboratory building, 46 by 46 ft., a ware-house, machine-shop, dry-house, office, high-pressure fire-system, complete electric system, and sewerage.

Refining Department.—It forms an extension, 105 by 108 ft., of the blast furnace building; contains two 80-ton softening furnaces, and four 50-ton desilverizing kettles, both served by the 20-ton crane of the blast furnace department; two 80-ton refining furnaces, 10 ft. below the kettles; and four merchant-kettles, from which the lead is raised by centrifugal pumps into Miller molding machines, Figs. 599 and 600. The zinc-silver crusts are treated in the silver-gold refinery, a brick building, 60 by 82 ft., which contains eight Faber du Faur retort furnaces, two cupelling furnaces, one silver-refining furnace, and a sulphuric-acid parting plant. The silver sulphate is decomposed with metallic copper, and the copper sulphate solution concentrated and crystallized in the copper sulphate building.

2. *Blank Silver-lead Smeltery.*—The outlines of this plant intended to treat 300 tons sulphide lead ore in 24 hr. are given in Figs. 316 and 317. The ore arrives on the upper track, is weighed and unloaded on to a belt conveyor which delivers it to the crushing and sampling division. The sampled ore is conveyed in a similar manner to the receiving bins of the oil-ignited Dwight-Lloyd 42-in. sintering machines, from which a conveyor-incline carries the blast roasted product to the sintered-ore bins. In line with these are the bins for coke and limestone. The charges, made up from the bins, are hauled to the feed-floor of the two blast furnaces, 42 by 128 in. at tuyères, which are similar to those of Northport, Wash., shown in Figs. 180–183. The lead bullion goes to the drossing plant; the cakes of matte are broken from a boom-derrick; the fore-hearths, hauled to the dump, are handled from an overhead crane; waste-slag cars are conveyed to the dump and poured; waste gases from the blast roasting division pass into a balloon flue and are cleaned in a Cottrell precipitating plant before they are carried off by the stack. The plan shows the loca-

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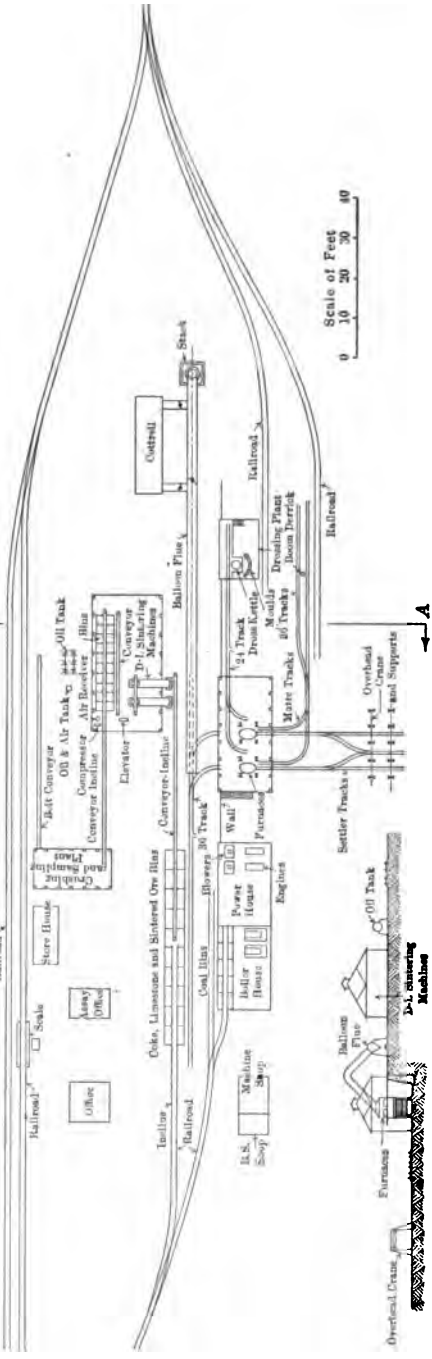

CROSS SECTION *A-A*

FIGS. 316 and 317.—Blank silver-lead smeltery.

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tions of the buildings for the power plant, machine- and blacksmith-shops, storehouse, oil- and air-tanks, general and assay offices. The plant was estimated in 1916 to cost, erected and complete, $268,735 or about $800 per ton per day. On account of the advance in cost of labor and materials, it is advisable to add from 10 to 20 per cent. to the estimate. The details of the estimate are as follows:

Smelting Department:		
2 furnaces same design as Northport Smelting and Refining Co.'s 42×128 in. at tuyères, 32-in. side-jackets, same height, etc., at 130,000 lb. ($11,500.00)	260,000 lb.	$23,000.00
4 settlers or fore-hearths, same design as Northport, at 16,000 lb. ($700.00).	64,000 lb.	2,800.00
18 matte trays at 2,500 lb	45,000 lb.	2,250.00
4 matte tray trucks at 900 lb	3,600 lb.	400.00
12 1¾-cu. ft. pots R. B. wheels for lead	4,800 lb.	720.00
2,000 lb. B.D. furnace steel	2,000 lb.	150.00
2 24-in. wind gates	1,100 lb.	180.00
Total weight and price f.o.b. factory	380,500 lb.	$29,500.00
Freight to New York		575.00
Packing		554.00
Erection		1,146.00
Building 55×120 ft		13,000.00
Total		$44,775.00
800 cu. ft. brick in each crucible (42,000)	210,000 lb.	$840.00
36,000 red brick for furnace above jackets	144,000 lb.	360.00
	354,000 lb.	$1,200.00
Erection		725.00
Total f.o.b. New York		$1,925.00
Sampling Plant—Building 40×78 ft.:		
1 10×20-in. jaw crusher, manganese	20,000 lb.	$1,200.00
1 set 36×16-in. rolls	31,500 lb.	2,450.00
3 36-in. Vezin samplers, single	3,600 lb.	600.00
1 conveyor between crusher and rolls, 16 in.×30 ft	3,000 lb.	300.00
1 conveyor between rolls and sampler, 16 in.×20 ft	2,000 lb.	200.00
1 B. & B. elevator 8 in.×36 ft	3,000 lb.	250.00
1 set 18×10-in. rolls	7,000 lb.	805.00
1 sample grinder No. 1	500 lb.	125.00
1 bucking plate and muller, 36×36 in	350 lb.	25.00
1 incoming belt conveyor, 20 in.×200 ft	4,000 lb.	2,000.00
1 outgoing belt conveyor, 16 in.×280 ft	5,000 lb.	2,800.00
Transmission	15,000 lb.	1,000.00
Belting	5,000 lb.	1,500.00
1 motor, 100 h.p.	5,600 lb.	1,600.00
Weight and price f.o.b. factory	105,550 lb.	$14,855.00
Freight to New York		170.00
Packing		208.00
Erection		756.00
Building 40×78 ft		4,500.00
Total		$20,489.00

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Power Plant, Building 55×85 *ft.:*

2 No. 7 Roots blowers	44,000 lb.	$2,710.00
2 engines, 105-h.p., Tangye automatic cut-off, Erie engine complete	17,000 lb.	1,800.00
1 blast pipe 60 in. diam.×140 ft	18,000 lb.	1,080.00
2 connections 24-in.×20-ft. blowers to pipe	2,000 lb.	150.00
2 24-in. blast gates	1,100 lb.	180.00
2 belts 12 in.×80 ft. each 6 ply	300 lb.	200.00
1 engine 11×15 in., Tangye automatic 70 h.p.	6,700 lb.	800.00
1 generator 220 volts, 50 k.v.a.	2,700 lb.	1,200.00
1 switchboard and instruments	2,300 lb.	550.00
Total f.o.b. factory	94,100 lb.	$8,670.00
Freight to New York		218.00
Packing		163.00
Erection		605.00
Building 55×85 ft.		9,000.00
Total		$18,656.00

Boiler Plant, Building 55×85 *ft.:*

3 boilers, 72 in.×16 ft., 125 h.p.	72,000 lb.	$4,200.00
1 B. F. pump, 5¼×3½×6 in.	3,000 lb.	150.00
Piping to engines	10,000 lb.	1,000.00
Brick for boilers:		
Red, 42,000	168,000 lb.	420.00
Fire-, 7,500	30,000 lb.	150.00
Cubic yards sand, 21		.....
Barrels cement, 21		40.00
Bushels lime, 100		25.00
Fire-clay, 4,200 lb.		10.00
Total f.o.b. factory	283,000 lb.	$5,995.00
Freight to New York		241.00
Packing		80.00
Erection		980.00
Building		9,000.00
Total		$16,296.00

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Drossing Plant, Building 40×60 ft.:		
1 kettle, 8 ft. diameter	9,000 lb.	$450.00
Iron work for setting	6,000 lb.	250.00
100 bullion molds	5,000 lb.	250.00
1 1½-in. centrifugal pump and 5-h.p. motor	1,000 lb.	350.00
1 chain block for same	500 lb.	50.00
Tools	1,000 lb.	100.00
Floor plates, cast iron	72,000 lb.	2,160.00
1 Dormant scale, No. 1046	2,000 lb.	125.00
Fire- and red-brick for kettle setting:		
Red, 40,000		400.00
Fire, 2,500		50.00
Cubic yards sand, 7		.....
Barrels cement, 7		12.00
Bushels lime, 35		8.00
Fire-clay, 1,400 lb		5.00
F.o.b. factory	96,500 lb.	$4,210.00
Freight to New York		146.00
Packing		24.00
Erection		511.00
Building		2,400.00
Total		$7,291.00
Sintering Plant, Building 70×140 ft.:		
2 D.L. sintering machines	100,000 lb.	$10,000.00
2 50-h.p. motors for fans	6,000 lb.	1,500.00
1 belt conveyor, 16 in.×105 ft	2,000 lb.	1,000.00
1 elevator, 12 in.×50 ft. B. & B	5,000 lb.	350.00
1 50-h.p. motor to drive conveyors, elevators, and sintering machines	3,000 lb.	750.00
10 belt feeders, 24×6 ft	10,000 lb.	1,250.00
10 bin hoppers and gates	10,000 lb.	500.00
2 sintered ore hoppers	10,000 lb.	500.00
1 air compressor belt drive, 8×8 in	4,000 lb.	750.00
1 oil tank, 8 ft.×24 in.×¼	8,000 lb.	400.00
1 10-h.p. motor for compressor	1,000 lb.	200.00
2 tanks, 24 in.×8 ft	500 lb.	100.00
Piping, valves and fittings	5,000 lb.	1,000.00
Transmission	20,000 lb.	1,000.00
Bolting	4,000 lb.	800.00
1 waste-gas flue, 5 ft. diameter, including connections, machines to fans	24,000 lb.	1,200.00
1 16-in. conveyor tripper over bins	1,000 lb.	150.00
1 16-in. conveyor tripper over s. machine	1,000 lb.	150.00
2 Evans speed cones	3,000 lb.	400.00
2 hoppers over s. machine, 6×6 ft	2,400 lb.	150.00
1 air receiver, 36×96 in.	1,200 lb.	160.00
F.o.b. factory	221,100 lb.	$22,310.00
Freight to New York		523.00
Packing		260.00
Erection		2,055.00
Building, 70×140 ft. including bins		20,000.00
Total		$45,148.00

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1 *Office Building*, 40×40 *ft*		$1,000.00
Office equipment		500.00
Total		$1,500.00
1 *Assay Office*, 30×40 *ft*		$800.00
Assay office equipment		2,000.00
Total		$2,800.00
1 *Store Room*, 25×60 *ft*		$1,000.00
1 Dorman W. scale, 1,046	2,000 lb.	125.00
Office equipment		100.00
Total		$1,225.00
1 *Machine shop*, 40×30 *ft*		800.00
1 lathe, 24 in.×10 ft		1,200.00
1 planer, 36×36 in		3,000.00
1 bolt cutter up to 1½ in		800.00
1 grindstone	1,000 lb.	40.00
1 drill press		1,000.00
1 blacksmith-shop, 20×30 ft		400.00
1 forge and bellows		75.00
1 anvil, 60 lb	60 lb.	15.00
1 set B.S. tools	1,000 lb.	100.00
Total		7,430.00
Freight		100.00
Packing		50.00
Erection		150.00
Total		$7,730.00
Yard:		
2 charging cars	14,000 lb.	$1,200.00
2 electric locomotives		6,000.00
1 yard crane, hand operated		1,500.00
1 support for same		2,000.00
Iron work for jib crane		250.00
Tracks and switches in front of furnace		500.00
1 RR. track scale		1,500.00
Tracks and switches. Bins to furnace		500.00
10 weighing hoppers for bins		3,000.00
1 conveyor 280 ft. long for sintered ore		6,000.00
1 balloon flue and supports, 12×400 ft	175,000 lb.	10,500.00
1 stack, 8×150 ft	40,000 lb.	2,000.00
Total f.o.b. factory		34,050.00
Freight to New York		750.00
Packing		200.00
Erection		4,000.00
Total		$39,900.00

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1 *Cottrell System*	$35,000.00
Water System for Fire, Granulating, and Domestic Use	5,000.00
Sewerage	2,000.00
Grading, Foundation, and Retaining Walls	10,000.00
RR. Tracks, Switches, and Trestles	5,000.00
Lighting System, Including Lamps, Wire, Poles, etc.	3,000.00
Tools and Equipment for Construction Work	1,000.00
Summary:	
Smelting department	$44,775.00
Smelting department, furnace brick	1,925.00
Sampling plant	20,489.00
Power plant	18,656.00
Boiler plant	16,296.00
Drossing plant	7,291.00
Sintering plant	45,148.00
Office building, etc	1,500.00
Assay building, etc	2,800.00
Store house	1,225.00
Machine- and boiler-shop	7,730.00
Yard equipment	39,900.00
Cottrell system	35,000.00
Water system	5,000.00
Sewerage system	2,000.00
Grading, foundations, etc	10,000.00
RR. tracks, etc	5,000.00
Lighting system	3,000.00
Tools, etc	1,000.00
Total	$268,735.00

123. Fuels.—The fuel used almost exclusively in the blast furnace is coke, as it holds up the charge, burns before the tuyères mainly to CO_2, and concentrates at the tuyère-region the heat generated. Formerly charcoal was the leading fuel, later came a mixture of coke and charcoal. Part of the coke has been replaced by bituminous coal, anthracite, wood, oil, and natural gas.

124. Coke.—A coke to be suited for the lead blast furnace ought be strong and hard enough to bear the burden, sufficiently porous to permit a good tonnage with a blast-pressure of not more than 3 lb. per. sq. in., and low in ash-content. Most kinds of coke have an adequate strength and hardness not to be crushed or abraded in the passage through the furnace. As regards porosity, gas-coke might form a suitable fuel, provided the pores are not too large, as they become more or less clogged and cause the formation of lumps of glazed coke which are not only useless, but in reality harmful. A porous coke burns quickly, but favors the creeping up of heat. In a number of cases a change from a porous to a denser coke has resulted in the disappearance of overfire. A very dense coke suited for the iron blast furnace working with a blast-pressure of 15 ± lb. per sq. in., burns slowly in a lead blast furnace and gives unsatisfactory reduction. A coke, therefore, which combines strength with a moderate degree of porosity best meets the requirements.

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Physical standards for lead blast-furnace coke have not been formulated as yet. A cubic foot of good coke piled loosely weighs about 27.5 lb.; it has about 50 per cent. of cell-space, and an ultimate compressive strength of 800–1000 lb. per sq. in. Coke from by-product ovens, which is harder than that from the bee-hive, has to be used in smaller pieces than bee-hive coke, as large pieces burn too slowly in the lead blast furnace.

The amount of fixed carbon of coke usually increases with the hardness, and this with the time and temperature given to the operation of coking. The percentage of volatile hydrocarbon is low, usually inside of 2 per cent. Coke with over 2 per cent. V.H.–C. has been noticed to favor creeping up of heat in the furnace. The amount of ash varies from 10 to 20 per cent.; it is usually nearer the first figure. Adding 2 per cent. V.H.–C. to an ash-content of, say, 12 per cent., gives by difference the F.C. at 86 per cent. Table 58 gives the proximate analyses of some important brands of blast furnace coke.

TABLE 58.—PROXIMATE ANALYSES OF COKE USED IN LEAD DISTRICTS

	Connellsville, Pa.	Cardiff, Wales	Grand River, Colo.	El Moro, Colo.	Crested Butte, Colo.	Pocahontas, Va.
Fixed carbon	87.46	95.00	93.75	87.47	92.03	92.550
Moisture	0.49	0.01	.....	.....	.....	0.347
Ash	11.32	4.26	5.49	10.68(*a*)	6.62(*a*)	5.749
Sulphur	0.69	0.68	0.76	0.85	.....	0.597
Phosphorus	0.029	0.02	0.10	.....	.....	.....
Volatile matter	0.011	.....	.....	1.85(*b*)	1.35(*b*)	0.757
Reference	(*c*)	(*c*)	(*c*)	(*d*)	(*d*)	(*f*)

In Table 59 are given analyses of coke-ash.

TABLE 59.—ANALYSES OF COKE-ASH

	Connellsville, Pa.	Starkville, Colo.	Grand River, Colo.	El Moro, Colo.	Crested Butte, Colo.	South Park, Colo.	Belt, Mont.	Horr, Mont.
SiO_2	44.64	65.45	50.1	84.50	42.25	29.10	51.6	57.0
Al_2O_3	25.12	25.60	.....	8.40	23.07	23.10	33.6	21.2
Fe_2O_3	22.73	8.00	14.3	7.10	26.43	47.80	10.0	19.0
CaO	6.95	.....	.....	.....	3.50	.....	.....	3.0
MgO	1.91	.....	.....	.....	1.40	.....	.....	.....
Reference	(*d*)	(*h*)	(*h*)	(*e*)	(*g*)	(*e*)	(*i*)	(*i*)

(*a*) Generally higher. (*b*) Including moisture. (*c*) "Mineral Resources of the United States," 1887, p. 396. (*d*) "Tenth Census of the United States," 1880, XV, p. 72. (*e*) Emmons, "Geology and Mining Industry of Leadville, Colo.," p. 642. (*f*) *Tr. A. I. M. E.*, 1890–91, XIX, 1034. (*g*) Private communication by C. H. Livingstone. (*h*) Private communication by M. W. Iles. (*i*) *Tr. A. I. M. E.*, 1904, XIV, 289.

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Table 59 shows that the leading constituents of coke-ash are SiO_2 and Al_2O_3; some varieties contain considerable amounts of Fe_2O_3. In computing a charge, the ash of the coke has to be taken into account.

Before coke is fed into the furnace, all the fines have to be removed, as they choke the charge, favor the formation of accretions, and, not burning freely, have to be carried out of the furnace by the slag. With more or less glazed particles of fine coke the slag is likely to become dirty and to retain particles of matte. The separation of fines is done with a coke-fork having prongs ¼ in. apart, while removing the coke from the shed. Sometimes a scoop is used, and the fines are screened out by dumping the coke over a grizzly which discharges the coarse coke on the feed-floor and the fines into a bin, whence they are removed, usually to be burned under a boiler, occasionally in a gas-producer or other places where their heat value can be utilized. Dwight[1] says that a reasonable amount of fines, such as usually accompanies a good quality of coke, does no harm, and he condemns their discard by forking.

125. Charcoal.—As regards porosity, charcoal is the best fuel for a lead furnace, as it consists[2] of a large number of small cells joined to each other by porous walls. Hence, being readily oxidized, it is a good reducing agent for oxidize ores, and does not require a high blast, which is an advantage. Its greater porosity causes greater bulk than coke (3 : 1), thus making the charge looser, which is favorable for quick smelting; further, its percentage of ash is low. The great disadvantage of charcoal is that few kinds can bear any heavy burden; it breaks up and crumbles. Fine charcoal is not only worthless as a fuel, but it is a bad conductor of heat. It makes unclean slags and also causes loss in metal by increasing the amount of flue-dust; hence nobody uses charcoal alone unless absolutely forced to do so. Further, the smelting zone tends to creep up and thus increase the loss of lead by volatilization.

Nut-pine (piñon) charcoal is the best, but it has to be well burned. Charcoal from lighter woods, such as yellow and white pine, quaking aspen, and cotton-wood can not be used alone in the furnace, and even with coke only a small percentage is allowable; some metallurgists condemn it entirely. Mesquite makes a good charcoal, but it is obtained with difficulty in large pieces. Charcoal from hard woods, such as mahogany, cedar, and oak, decrepitates in the furnace. When charcoal is exposed for any length of time to the open air it breaks up and the amount of fines becomes large. On the other hand, its quality is said to be improved by storing, through the oxygen that it absorbs. Lead smelters do not like to have large amounts of charcoal on hand. It should be stored where it is not exposed to the sun. Charcoal has in a few instances been replaced by an equivalent in reducing power of split wood cut up into 12-in. lengths. The wood keeps the charge open and the charcoal formed arrives less crushed at the tuyère-region than if it had been charged as charcoal.[3]

[1] *Tr. A. I. M. E.*, 1902, XXXII, 365.

[2] Thörner, *Stahl und Eisen*, 1886, VI, 71.
Hofman, "General Metallurgy," 1918, p. 208.

[3] Hofman, "Metallurgy of Copper," 1918, p. 182.

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Light-wood charcoal contains about 2 per cent. of ash. As this consists principally of alkali and alkali-earth carbonates, it does not affect a charge to any appreciable extent. One bushel weighs about 14 lb. The height of charge it can bear varies too much with the different kinds of charcoal to give a general figure; in all cases it is less than with coke.

126. Coke and Charcoal.—From what has been said it is obvious that the ideal fuel for the lead smelter must combine the strength of coke and the porosity of charcoal; thus, at least with a low-pressure blast, a mixture of coke and charcoal will put through more charges in a given time than either alone. The coke bears up the charge and prevents the charcoal from being crushed. This burns quickly, helps to ignite the coke, and, having hardly any ash, leaves hollow spaces for the blast to penetrate. At present, when furnaces are run with a pressure of from 2 to 3 lb. to the square inch and oxide ores form a small percentage of the charge, charcoal is never used. The greater height of furnace and with it of charge necessitated by the increased volume of air blown into the furnace, would cause the charcoal to be broken; the fine charcoal would cause the disadvantages given in §125; it would burn near the top of the charge and cause overfire.[1]

127. Coke and Bituminous Coal.—Neill[2] succeeded in 1891 in replacing part of the coke by a non-caking or only slightly caking bituminous coal, using separately lump, nut, and pea coal. He gives as results of his experiments that, beside the direct saving in substituting the cheaper bituminous coal for the coke, jackets, slag, and lead appeared hotter, the tuyères brighter, and the crucible kept open better. The slag assays ran lower in values than with the usual coke and charcoal mixture, and the separation of slag and matte was good. On the feed-floor the charges settled more evenly, as fewer zinc accretions were formed, and the top was cooler than usual. While the volume of smoke was larger, there was no greater loss in the flue-dust on account of the charge being cooler. In his furnace, 36 by 78 in. at the tuyères, and 12 ft. from the tuyères to the charging-door, he was using coke and charcoal in the proportion of 3 : 1, and was able to have 27 per cent. of bituminous coal in his fuel charge, replacing an equivalent amount of coke. He expected that with a higher furnace 50 per cent. of his fuel might be bituminous coal.

The example of Neill in using mixtures of coke and bituminous coal, instead of coke or coke and charcoal, has been followed by others and proved successful. Austin[3] gives a summary of 18 months' experience in the use of mixed fuels—coke, charcoal, and bituminous coal—in the blast furnace (having a 12-ft. working height) of the Germania Lead Works, Salt Lake City, Utah. Table 60 contains the leading data:

[1] Bretherton, *Tr. A. I. M. E.*, 1915, LII, 730.
[2] *Op. cit.*, 1891, XX, 165.
[3] *Eng. Min. J.*, 1894, LVIII, 558.

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TABLE 60.—ANALYSES OF BLAST FURNACE FUELS

Blast furnace fuel	Cost per ton	Character	Per cent. ash	Composition of ash, per cent.				Calorific power: gm. lead reduced by 1 gram fuel
				SiO_2	FeO	CaO	Cu	
Coke from Castle Gate, Utah	$7.50	Friable	14.0	35.0	25.7	6.5	2.6	24.5
Coke from Crested Butte, Colo.	11.00	Fairly hard	11.3	32.2	30.4	6.6	2.8	28.5
Nut coal from Castle Gate, Utah.	3.00	Friable	8.4	42.0	20.8	10.0	....	24.0
Charcoal	14.00	.....	.....	....	....	....	....	.....

Blast furnace fuel	Coke: coal			Coke: coal			Coke: charcoal: coal		
	Proportions in mixture	Cost per ton charge	Per cent. of mixture	Proportions in mixture	Cost per ton charge	Per cent. of mixture	Proportions in mixture	Cost per ton charge	Per cent. of mixture
Coke from Castle Gate, Utah	14:2	$1.33	16	12:4 9:9	$1.04 0.94	16 18	9:2:6	$1.07	17
Coke from Crested Butte, Colo.	14:2	1.82	16	12:5 9:9	1.47 1.26	17 18	.....	.....	.....
Nut coal from Castle Gate, Utah	.....	.....	.....	.....	.....	.....	.....	.....	.....
Charcoal	.....	.....	.....	.....	.....	.....	.....	.....	.....

The conclusions drawn by Austin are, that in replacing charcoal and some of the coke by bituminous coal, the amount of fuel required to the ton of charge has to be increased by 1 or 2 per cent.; that the effect is similar to that of charcoal in promoting the running of the furnace, but that an excess of bituminous coal is likely to choke up the charge and can be corrected by substituting charcoal; that bituminous coal appears to work unfavorably with coppery and arsenical ores, the crucible filling up slowly; that as regards manipulation the feeding of bituminous coal must be stopped whenever yellow smoke issues from the surface of the charge; that overfire is to be avoided, as it ignites the explosive gas-mixture on its passage into the downcomer; and that lastly the blast may never be taken off entirely (at least one blower must be kept going), as the same gas-mixture coming in contact with air in the furnace will be ignited by the hot charge. The replacing of charcoal and coke by bituminous coal is recommended, if the latter is considerably cheaper, and if copper and arsenic are present in small enough quantities to permit its use.

128. Coke and Anthracite.—The leading disadvantages of using anthracite in a furnace are its denseness, which causes it to burn slowly; and its property of decrepitating when heated, which gives rise to dirt troubles caused by infusible

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and non-combustible mixtures of fine coal and ashes. Dwight[1] publishes some experiments made by Von der Ropp in substituting anthracite of goose-egg size for part of the coke, no charcoal being used. The furnace was 36 by 80 in. at the tuyère-level and 9 ft. from there to the feed-floor. The result was that the smelting power of the furnace was reduced as the proportion of anthracite was increased, *e.g.*, one-third with anthracite as 60 per cent. of the fuel; otherwise the furnace remained in good condition. The top kept cool and the crucible open; there was a good reduction shown by a clean slag (0.4 to 0.8 per cent. Pb), a matte low in lead (8 per cent. Pb, 4 per cent. Cu), and a speiss having a coarsely crystalline structure. Finally, less zinc accretions were formed than when coke alone was used, the charge containing 7.5 per cent. Zn.

129. Liquid and Gaseous Fuel.—The use of crude oil, injected at the tuyère-level, to replace part of the normal amount of coke, has been suggested, but not tried, at least with lead blast furnaces. In matting copper ores, success has been claimed for the use of oil.[2]

Gaseous fuel has been used in a single instance. At the works of the Pennsylvania Lead Company[3] Blake introduced natural gas with the blast by inserting a gas-pipe through the tuyère-pipe. The amount of natural gas was regulated by stop-cocks and the blast-volume increased so as to supply sufficient air for the combustion of the gas. Thirty per cent. of coke was successfully saved in this way. By replacing 60 per cent., the top of the furnace became too hot. That solid fuel cannot be entirely replaced by gaseous is clear from the reactions going on in a blast furnace which require solid carbon. For pecuniary reasons it is improbable that any artificial gaseous fuel will ever be used in the blast furnace.

130. Weight of Fuel.[4]—The weight of fuel required in a lead blast furnace is generally expressed in terms of percentage of the total weight of the charge (ore, flux, and slag). Some metallurgists[5] prefer to measure the individual charges of coke going into the furnace instead of weighing them, as the greater or smaller amount of water contained in the coke is likely to give wrong weights. They weigh given amounts of dried coke into the charging larrie, mark the heights to which these reach, and are then guided by the markings in feeding the furnace.

The simple statement that a furnace is run with a definite percentage of coke is not sufficient to convey a clear idea of how much heat is being used, as

[1] *Tr. A. I. M. E.*, 1891, XX, 169.

[2] Hamilton, *Eng. Min. J.*, 1911, XCI, 244.
Kiddie, *op. cit.*, 1911, XCII, 434.
Waters, *op. cit.*, 1912, XCIII, 877.
Lang, *Min. Sc. Press*, 1913, CVI, 298.
Marquard, *Canad. Min. J.*, 1915, XXXVI, 472.

[3] *Tr. A. I. M. E.*, 1886–87, XV, 661.

[4] Austin, *Min. Sc. Press*, 1907, XCIV, 341.
Ralli, *Rev. Un. Min.*, 1911, XXXIV, 313, XXXV, 1.
von Schlippenbach, *Metall. u. Erz.*, 1915, XII, 403.

[5] von Schlippenbach, *loc. cit.*

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the percentage of ash in different kinds of coke varies greatly. In recent years it has become the custom to add the percentage of fixed carbon to the figure denoting the percentage of coke. A furnace is run with 13 per cent. coke which, with 86 per cent. F. C., corresponds to 11 per cent. C. Misunderstanding occasionally arises from deducting the pounds of lead contained in the charge and referring the percentage of fuel only to the slag and matte material. Often the percentage of fuel used refers to the weight of the charge excluding the slag that is added, for the reason that an addition of slag, because it facilitates the smelting, does not within ordinary limits call for extra fuel.

In general it may be said that a furnace requires from 12 to 15 per cent. coke of the total weight of ore plus flux to run satisfactorily; this figure, with coke of 86 per cent. C, corresponds to 10.5–13.0 per cent. C. A charge with metallic sulphides requires less fuel than one that is made up of oxides. A good rule[1] is to avoid any excess of fuel and to keep on the "ragged edge of reduction," as this gives large tonnage, long campaigns, good furnace conditions, and good metallurgical work; the slag will assay 0.5–1.0 per cent. Pb, and be fluid and hot; the volume of smoke issuing from the throat will be small, and gray or dark in color; and the lead output will be good. Over-reduction, *i.e.*, using too much fuel, is accompanied by no end of disturbances. There is danger of iron and copper sows; of making much speiss in the presence of arsenic; the crucible-content becomes sticky; the furnace tightens and speed is lessened; slags run slowly although they are hot; coke appears at the tap-hole, and white smoke and overfire at the top; slags contain too little lead, viz., 0.3–0.4 per cent., and show sparks of burning coke. In under-reduction, *i.e.*, using too little fuel, the speed is high; the slags, while fluid, are cold and look heavy, and run 1 per cent. Pb and over; a large volume of thick, grayish white smoke arises from the throat; and the yield in lead is bad. The correction is made by adding, say with a slag of 2 per cent. Pb, 4 per cent. more coke, and then reducing this gradually when the results of the correction have been noticed. If the fuel is to be reduced when the furnace is doing fair work, it is wise to decrease by not more than 1 per cent. at a time. In running a new charge, it is advisable to use a little extra fuel in order that the assays and physical appearances may form clear guides for making any necessary changes.

The amount of C theoretically required to smelt a charge can be calculated from the thermal balance sheet (§156). In practice the amount of coke required varies with its character, with the fusibility of the charge, with the time of year, with the altitude at which the smelting is carried on, and the pressure of the blast.

As regards the character of the fuel, coke that is rich in ash is not only an inferior fuel in proportion to the smaller amount of C it contains, but a considerable quantity of this C is consumed to melt the ash and the fluxes necessary to slag it. For this reason a smaller amount of charcoal than of coke would seem to be required. The exact opposite, however, is the case, as the charcoal crumbles, is crushed in its descent in the furnace, burns more to CO at the tu-

[1] Austin, *loc. cit.*

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yères than the denser coke, and thus produces less heat, which has to be made up by additional fuel, and is oxidized more quickly by the CO_2 of the ascending gases than is the coke.

The richer an ore is in lead the less fuel will be needed, as the amount of gangue to be smelted is correspondingly smaller; also the more fusible will be the rest of the charge. An ore containing zinc requires more fuel than one that is free from it; a calcareous slag requires more fuel than one that is furruginous; a coarse and open charge requires less fuel than one that is fine and dense, as fine ore causes slow running.

In summer less fuel is generally required than in winter, not so much owing to the higher temperature, which may be counted as balanced by the larger percentage of moisture in the air, as to the more rapid evaporation of the water contained in ore, flux, and fuel. The difference may be as much as 5 per cent. Furnaces run better at night than during the day, as there is less moisture in the air.

The altitude at which an ore is smelted makes a great difference in the amount of fuel required. Hahn,[1] for instance, in 1881, gave the figures of 14 and 17 per cent. in Salt Lake City (4000 ft. above the level of the sea) as against 22 and 24 per cent. in Leadville (10,000 ft.); the lower figure refers to the summer, the higher to the winter season. A partial explanation of this fact was given by Headden at a meeting of the Colorado Scientific Society, in that a cubic foot of air entering the blast furnace under a certain pressure will expand more at a high elevation, where the air is rarefied, than at sea-level, and that consequently more heat will be consumed which has to be made up by an extra amount of fuel. This accounts, however, only for a small part of the extra heat required. Palmer[2] has shown that the difference in fuel consumption at high in comparison with low elevations is due to the fact that the velocity of combustion is directly proportional to the concentration of the O in the air entering through the tuyères. As the temperature at the tuyères is proportional to the rate of combustion of the C, the low temperature caused by the slow burning at a high elevation has to be corrected by adding more fuel to the charge.

Lloyd[3] records that in matting sulphide-copper ore at an elevation of 14,000 ft. the smelting power was lower and the pyritic effect smaller than at sea-level; he also states, as does Sacio,[4] that heat losses by radiation have to be avoided as much as possible. The statements of both give additional evidence of the slower rate of oxidation at a high than at a low altitude. Similar experience has been had in roasting pyrite concentrates at different elevations.

The pressure of the blast has a great influence on the fuel consumption. With the pressure of ¾ and 1 lb., common in the early eighties, the amount of coke needed at an elevation of 5000 ft. was about 15 per cent., and at 10,000 ft. 18

[1] *Min. Res. U. S.*, 1882, p. 399.
[2] *Eng. Min. J.*, 1906, LXXXI, 134.
[3] *Tr. Met. Inst. Min. Ind.*, 1909–10, I, 11.
[4] *School Min. Quart.*, 1913, XXIV, 344; *Met. Chem. Eng.*, 1913, XI, 499.

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per cent., and as much as 22 per cent., if a mixture of coke and charcoal was used. This has fallen with a pressure of from 2 to 3 lb. to 12 and 15 per cent. respectively. The reason for this saving is partly because of the smaller amount of carbonate ore treated, but mainly on account of the more even and perfect combustion of the coke at the region of the tuyères, because the blast comes in better contact with the fuel. If, however, the pressure be strong enough for the

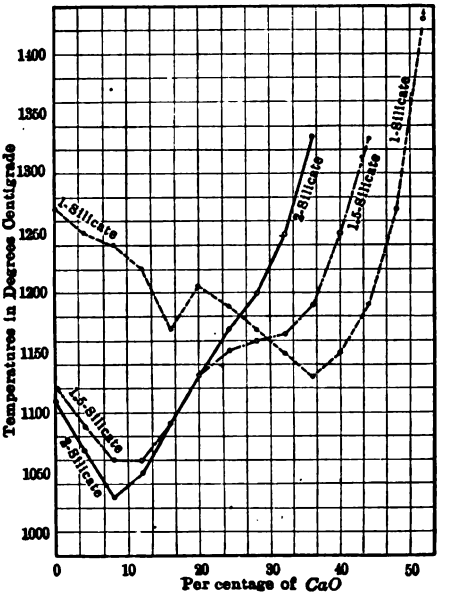

FIG. 318.—Formation temperatures of some ferro-calcic singulo-silicates.

air to penetrate too much into the coke the C will burn only to CO, with the consequent reduction of temperature; the porosity of the coke must therefore be a gauge for the blast.

With charcoal, where high-pressure blast is not permissible, the older figures hold good to-day, as much as 26 and 28 per cent. being used, according to Hahn.[1]

131. Lead Slags, Thermochemical Properties.[2]—The slags made in a lead blast furnace are mixtures mainly of ferrous and calcic silicates; the parts played

[1] *Min. Res. U. S.*, 1882, 339.

[2] Fulton, C. H., "Principles of Metallurgy," McGraw-Hill Book Co., New York, 1910, p. 244.

Hofman, H. O., "General Metallurgy," McGraw-Hill Book Co., New York, 1918, p. 433.

Hahn, *Min. Res. U. S.*, 1882, p. 325.

Guyard in Emmons, "Geology and Mining Industry of Leadville, Colo.," monograph, XII, U. S. Geol. Survey, p. 701.

Iles, *Min. Res. U. S.*, 1883–84, p. 440.

Schneider, *Tr. A. I. M. E.*, 1882–83, XI, 56.

Keller, *Proc. Instit. Civ. Eng.* (England), 1893, CXII, 203.

Struthers, *School Min. Quart.*, 1894–95, XVI, 356 (partial bibliography).

Henrich, *Eng. Min. J.*, 1883, XXXVI, 211.

Dwight, Mathewson, Boggs, *Tr. A. I. M. E.*, 1916, LVI, 627–629.

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by other bases will be considered in §136 and following. The freezing-point curves, Fig. 318, of the singulo-silicate series, $2RO.SiO_2$, the sesqui-silicate, $4RO.3SiO_2$, and the bi-silicate, $RO.SiO_2$, show that these silicates are eutectic mixtures; the existence of chemical compounds has not been definitely proved although there is at SiO_2 31.07, FeO 48.93, CaO 20 per cent. an indication in the singulo-silicate curve of the compound $2(2FeO.SiO_2).1(2CaO.SiO_2)$, which contains SiO_2 31.03, FeO 49.65, CaO 19.31 per cent. The more recent work of Zmiel,[1] dealing with ferrous silicates, and of Konstantinoff and Selivanoff with ferro-calcic bi-silicates[2] shows the general V-shaped form of the freezing-point curve with the absence of a chemical compound, while that of Selivanoff[3] dealing with ferro-calcic singulo-silicates points to the existence of the chemical compound $2(2FeO.SiO_2).3(2CaO.SiO_2)$ freezing at about 1250° C. with an eutectic on either side, that on the FeO-side melting at about 1150° C. and that on the CaO-side at about 1210° C.

The formation temperatures of the singulo-silicates of Fig. 318 are given in Table 61. They show that with the replacement of FeO by CaO the formation temperature of the ferrous silicate of 1270° C. falls to 1170° with the slag containing 16 per cent. CaO, it rises to 1205° with 30 per cent. CaO; falls again with

TABLE 61.—FORMATION TEMPERATURES OF FERRO-CALCIC SINGULO-SILICATES

Chemical composition of slag			Melting-point, degrees C.
SiO_2, per cent.	FeO, per cent.	CaO, per cent.	
29.20	70.80	0	1,270
29.75	66.25	4	1,250
30.09	61.91	8	1,240
30.42	57.58	12	1,220
30.76	53.24	16	1,170
31.07	48.93	20	1,205
31.40	44.60	24	1,190
31.70	40.30	28	1,170
32.10	35.90	32	1,150
32.30	31.70	36	1,130
32.70	27.30	40	1,150
33.10	22.90	44	1,190
33.44	18.56	48	1,270
33.79	14.21	52	1,430+

further replacement until it reaches the lowest point at 1130° with 36 per cent. CaO; and then rises steadily, probably to the formation temperature of $2CaO.SiO_2$, which lies at 2080° C.

The formation temperatures of the sesqui-silicates are given in Table 62. The ferrous sesqui-silicate has a lower formation temperature (1120° C.) than

[1] *Rev. Mét. Extr.*, 1913, X, 611.
[2] *Op. cit.*, 1913, X, 607.
[3] *Op. cit.*, 1915, XII, 309.

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the singulo-silicate (1270°); it also falls with the replacement of FeO by CaO, but reaches its minimum when the slag contains 12, or perhaps 8 per cent. CaO.

TABLE 62.—FORMATION TEMPERATURES OF FERRO-CALCIC SESQUI-SILICATES

Chemical composition of slag			Melting-point, degrees C.
SiO_2, per cent.	FeO, per cent.	CaO, per cent.	
38.46	61.54	0	1,120
38.90	57.10	4	1,090
39.34	52.66	8	1,060
39.78	48.22	12	1,060
40.22	43.78	16	1,090
40.66	39.34	20	1,130
41.11	34.89	24	1,150
41.54	30.46	28	1,160
41.99	26.01	32	1,165
42.42	21.58	36	1,190
42.87	17.13	40	1,250
43.31	12.69	44	1,330+
43.75	8.26	48	.....
44.19	3.81	52	.....

The formation temperatures of the bi-silicates are given in Table 63.

TABLE 63.—FORMATION TEMPERATURES OF FERRO-CALCIC BI-SILICATES

Chemical composition of slag			Melting point, degrees C.
SiO_2, per cent.	FeO, per cent.	CaO, per cent.	
45.45	54.55	0	1,110
46.00	50.00	4	1,070
46.53	45.47	8	1,030
47.04	40.96	12	1,050
47.56	36.44	16	1,090
48.02	31.98	20	1,130
48.57	27.43	24	1,170
49.19	22.81	28	1,200
49.60	18.40	32	1,250
50.11	13.89	36	1,330
50.63	9.37	40	1,430
51.14	4.86	44	.....
51.65	0.35	48	.....
51.73	0.00	48.27	.....

The temperature at which the ferrous bi-silicate forms, 1110° C., is lower than that of the sesqui-silicate, 1120° C.; it is also lowered with replacements of FeO by CaO, and reaches its lowest point with a slag containing not more than 8 per cent. CaO.

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The formation temperatures of ferro-calcic silicates are therefore the lower, the higher the silicate-degree; a slag with a high silication reaches its minimum formation temperatures with less CaO than one in which it is low.

The main slag for the lead blast furnace is the singulo-silicate; in most cases, however, it is made slightly more silicious than called for by the formula. Calculation of the formula from the chemical analysis[1] often shows that a slag is made up of ferrous singulo-silicate ($2FeO.SiO_2$) and calcic bi-silicate ($CaO.SiO_2$).[2] Of the many possible mixtures of ferrous and calcic singulo-silicates, certain definite proportions of SiO_2, FeO, and CaO, have been found to work satisfactorily in the lead blast furnace; they have become representative for the work. According to Eilers[3] such a *Typical Lead Slag* should not contain over 0.75 per cent. Pb or 0.5 oz. Ag per ton, provided the lead bullion does not run higher than 300 oz. per ton; nor have a density over 3.6; nor permit either accretions in the hearths (thus keeping red-hot the lead in the crucible), or any creeping up of heat. The lead- and silver-contents of slags made at present are higher, as the ores treated are not so pure as those smelted in the early eighties.

Table 64 gives such typical slags. The compositions of the slags have been calculated to total 90, leaving 10 per cent. for other components, such as Al_2O_3, ZnO, alkali, etc. In smelting ores that contain few of these other components the sum of SiO_2, FeO, and CaO will often total 95 or 96 per cent.; if, however, this is not the case, the sum of the three main constituents may be only 85 or 86, and even less, as shown in §145 in connection with zinc, and as found in smelting ores from Cripple Creek, Colo., running high in Al_2O_3.

TABLE 64.—TYPES OF LEAD SLAGS

Authority	Type	Special name	SiO_2	Fe(Mn)O	Ca(MgBa)O	Total
Eilers	C	Quarter slag	28	50	12	90
LaPise-Eilers	E	Half slag	30	40	20	90
Schneider	F	Three-quarters slag	33	33	24	90
Raht	G	One-to-one slag	35	27	28	90
Hahn	H	Silicious quarter slag	32	47	11	90
Page	I	Silicious half slag	35	38	17	90
Livingstone	J	Half slag	31	38	21	90
Hahn	K	Silicious half slag	34	37	19	90
Murray	M	Three-quarters slag	36	31	23	90

The special names given in column 3 of Table 64 are based upon the ratios of FeO and CaO; thus slag of type C is called a quarter slag on account of FeO: CaO = 4 : 1 approximately. This nomenclature originated in the idea of slags being chemical compounds; it has lost its supposed value, but is given as it is still much in vogue.

[1] Graphical representation by Liddell, *Eng. Min. J.*, 1914, XCVII, 318. Deming, *J. Ind. Eng. Chem.*, 1916, VIII, 269.

[2] Table for such slags, see Hofman, H. O., "Metallurgy of Copper," McGraw-Hill Book Co., New York, 1918, p. 200.

[3] *Eng. Min. J.*, 1881, XXXI, 246.

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In Table 65, calculated by Wm. Allen Smith of Herculaneum, Mo., are given some types of lead slags which are similar to those of Table 64, and others which are different, but have been made with success in Texas, Colorado, and Utah. The totals of SiO_2, FeO, and CaO are seen to range from 77 to 92 per cent. and to include the pure slag in which they are equal to 100 per cent. In all the types it is striking that 1 lb. FeO requires for fluxing 0.557 lb. SiO_2; that 1 lb. CaO requires 0.714 lb. SiO_2; and that 1 lb. FeO has the same fluxing power as 0.78 lb. CaO. In order to form a singulo-silicate, 1 lb. FeO requires 0.416 lb. SiO_2, and 1 lb. CaO requires 0.535 lb. SiO_2; for a sesqui-silicate, 1 lb. FeO requires 0.625 lb. SiO_2, and 1 lb. CaO requires 0.803 lb. SiO_2. The silicate degree of the slags in Table 65 lies between that of a singulo- and a sesqui-silicate; in other words these types of slags contain a slight excess of SiO_2 over that called for by a singulo-silicate.

TABLE 65.—TYPES OF LEAD SLAG

Type	⅜			⅜ and ½			½			¾			⅞			1		
Total	SiO_2	FeO	CaO	SiO_2	FeO	CaO	SiO_2	FeO	CaO	SiO_2	FeO	CaO	SiO_2	FeO	CaO	SiO_2	FeO	CaO
100	37.5	45.0	17.5	37.65	43.30	19.05	37.8	41.6	20.6	38.5	34.6	26.9	38.65	32.80	28.55	38.8	31.0	30.2
77	28.9	34.6	13.5	29.0	33.3	14.7	29.1	32.0	15.9	29.7	26.6	20.7	29.7	25.3	22.0	29.9	23.9	23.2
78	29.3	35.1	13.6	29.4	33.8	14.8	29.5	32.4	16.1	30.0	27.0	21.0	30.1	25.6	22.3	30.3	24.2	23.5
79	29.6	35.6	13.8	29.7	34.2	15.1	29.9	32.8	16.3	30.4	27.3	21.3	30.5	25.9	22.6	30.6	24.5	23.9
80	30.0	36.0	14.0	30.1	34.6	15.3	30.2	33.3	16.5	30.8	27.7	21.5	30.9	26.2	22.9	31.0	24.8	24.2
81	30.4	36.4	14.2	30.5	35.1	15.4	30.6	33.7	16.7	31.2	28.0	21.8	31.3	26.6	23.1	31.4	25.1	24.5
82	30.7	36.9	14.4	30.9	35.5	15.6	31.0	34.1	16.9	31.6	28.4	22.0	31.7	26.9	23.4	31.8	25.4	24.8
83	31.1	37.4	14.5	31.3	35.9	15.8	31.4	34.5	17.1	32.0	28.7	22.3	32.1	27.2	23.7	32.2	25.7	25.1
84	31.5	37.8	14.7	31.6	36.4	16.0	31.8	34.9	17.3	32.3	29.1	22.6	32.5	27.5	24.0	32.6	26.0	25.4
85	31.9	38.2	14.9	32.0	36.8	16.2	32.1	35.4	17.5	32.7	29.4	22.9	32.8	27.9	24.3	33.0	26.3	25.7
86	32.2	38.7	15.1	32.4	37.2	16.4	32.5	35.8	17.7	33.1	29.8	23.1	33.2	28.2	24.6	33.4	26.7	25.9
87	32.6	39.2	15.2	32.7	37.7	16.6	32.9	36.2	17.9	33.5	30.1	23.4	33.6	28.5	24.9	33.7	27.0	26.3
88	33.0	39.6	15.4	33.1	38.1	16.8	33.3	36.6	18.1	33.9	30.4	23.7	34.0	28.9	25.1	34.1	27.3	26.6
89	33.4	40.0	15.6	33.5	38.5	17.0	33.7	37.0	18.3	34.3	30.8	23.9	34.4	29.2	25.4	34.5	27.6	26.9
90	33.8	40.5	15.7	33.9	39.0	17.1	34.0	37.4	18.6	34.6	31.1	24.3	34.8	29.5	25.7	34.9	27.9	27.2
91	34.1	41.0	15.9	34.3	39.4	17.3	34.4	37.8	18.8	35.0	31.5	24.5	35.2	29.8	26.0	35.3	28.2	27.5
92	34.5	41.4	16.1	34.6	39.9	17.5	34.8	38.3	18.9	35.4	31.8	24.8	35.5	30.2	26.3	35.7	28.5	27.8

It is held by W. A. Smith that as long as the SiO_2-content of a slag has been neutralized by FeO and CaO in the amounts given, the slag will run successfully. Of course, the presence of other constituents, as for example of ZnO or Al_2O_3, will restrict the choice in making a high-FeO or a high-CaO slag, with corresponding differences in the SiO_2-content.

132. Thermophysical Properties.—The fusibility of a slag, which always includes the formation temperature, depends upon the percentage of SiO_2 and the character of the bases; the effects of both will be taken up in §134 and following. The more fusible a slag, the larger will be the amount of charge smelted per unit of fuel. With the correspondingly low temperature prevailing, the reducing power of the furnace will be diminished; therefore a slag that is very readily fusible is not always desirable.

The liquidity of a slag depends largely upon its fusibility. Usually the free-running temperature lies 100° C. or more above the formation temperature.[1]

[1] Steffe-Hofman, *Min. Ind.*, 1908, XVII, 597.

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An interesting instance of a large difference between the two temperatures is shown by experience of Carpenter[1] in matting Cripple Creek gold ore at Florence, Colo. The presence of a large amount of alkali in the phonolite rock caused the formation of glassy slag at a temperature lying very much below that necessary for free-flowing, and the furnace froze. On pulling down the front, the furnace was found to be filled more or less with glass. An addition of MgO-bearing rock to the charge of the next run corrected the evil.

A slag ought to be sufficiently fluid to flow freely and to allow a perfect separation of lead in the furnace and of matte in the fore-hearth. A correctly composed slag, which would otherwise be fluid, becomes viscid, if the weather changes to wet or cold; in such cases more fuel is required. It is difficult, even for the practiced eye, to say always whether the viscidity comes from lack of fusibility or from a slight insufficiency in fuel. As a rule, singulo-silicate slags containing earthy and metallic bases solidify quickly without first becoming pasty. When the tap-hole has been closed, the slag, if good, will drop slowly into the pot, drawing a small thread as it leaves the spout; a slag with an excess of base will fall off quickly in little round drops. Most slags have certain characteristics in their manner of running, which have to be studied by observation. In close connection with the running of the slag from the furnace is the manner of its rising in the slag-pot, and the appearance of the surface when it has just solidified and is still red. Many slags show very characteristic surfaces. *E.g.*,[2] the slag type E, Table 64, with SiO_2 30, FeO 40, CaO 20 per cent. shows a slight thread when dropping from the spout. When rising in the pot, more or less concentric rings are seen, and radial lines (usually four) when the pot is about full.

The melting-points of lead blast-furnace slags lie nearer 1,200 than 1,100° C. Schertel[3] gives 1,030° C., Guyard[4] assumes 1,200°, Iles[5] finds 1,034°, Goetz (quoted by Iles) finds a range from 1,100 to 1,200°, Landis[5] gives 1,114°, Clevenger[7] finds in two instances the temperatures of slags flowing from furnaces to be 1,149 and 1,170° C.

The heats of formation from the constituent oxides[8] per kilogram of silicate show values ranging from 133 to 193 Cal. Additional data regarding singulo- and bi-silicates of iron and manganese have been found by Wolgodine.[9]

The specific heat between zero deg. C. and the melting-point, t, is $0.20185+0.0000302t$, and that at the melting-point $0.20185+0.000604t$.

[1] *Tr. A. I. M. E.*, 1900, XXX, 130.

[2] Austin, *Min. Sc. Press*, 1907, XCIV, 252.

[3] *Freiberg. Jahrb.*, 1880, p. 45.

[4] *Loc. cit.*, p. 744.

[5] *School Min. Quart.*, 1895–96, XVII, 20.

[6] Richards, J. W., "Metallurgical Calculations," McGraw-Hill Book Co., New York, 1918, p. 500.

[7] *Met. Chem. Eng.*, 1913, XI, 448.

[8] Hofman-Wen, *Tr. A. I. M. E.*, 1910, XLI, 511.

[9] *Rev. Mét., Extr.*, 1914, XI, 395.

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The total heat of fusion is given by Landis[1] as 302 Cal. and by Austin[2] as 300 Cal. per kg. Landis gives the heat content, solid at 1,114° C. (the melting-point) as 262 Cal., hence the latent heat of fusion is 40 Cal.

133. Physical Properties.—*Crystallization.*—Well-composed slags have a decided tendency to crystallize. The center of a cone of slag is generally more crystalline than any other part because the cooling has been slower. The slag, SiO_2 30, FeO 40, CaO 20 per cent.[3] from a big pot as well as from a ladle-sample will show crystals in the central part. An excess of FeO tends toward the formation of large crystals, whereas an excess of CaO favors a more granular structure. The crystalline character of the normal slag extends to the surface; if it stops short of the edge, and this has a reddish tinge, there is an excess of SiO_2; if the cavities in the central part show no crystals but are smooth and hard looking, there is an excess of FeO in regard to SiO_2; if the top of a big pot has a smooth surface, the slag contains an excess of CaO over the normal. Slags that crystallize usually become amorphous (glassy) if chilled suddenly, and crypto-crystalline if not given sufficient time to develop crystals. A small percentage of zinc oxide in the slag interferes greatly with the crystallization. Iles[4] thought at one time that the form in which a slag crystallized stood in some relation to the percentage of lime it contained. He said, Fig. 319, that slags with from 3 to 5 per cent. of CaO crystallize like *a*; with 8 to 12 per cent. like *a'*, with 15 to 18 per cent. like *b*, with 19 to 22 per cent. like *c*, with 23 to 25 per cent. like *d*, with 25 to 27 per cent. like *e*, with 30 to 35 per cent. like *g*. The forms *f*, *f'*, and *h* represent crystals formed in attempting to slag very silicious ores with lime alone. But the percentage of CaO can not be the only cause for the different forms of crystallization, as Raht[5] obtained from a single pot of slag three different forms of large, well-developed crystals of approximately the same composition, given in Table 66.

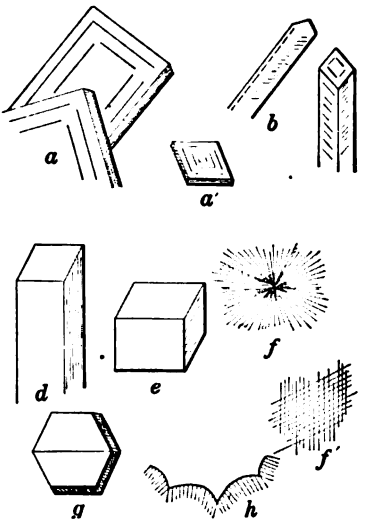

FIG. 319.—Crystal-forms of lead-slags.

It is to be noted that the same slag will melt at a lower temperature when glassy than when crypto-crystalline. This is due to the differentiation that occurs in cooling, crystals forming on the one hand, and ground mass on the other, both of which have different compositions and different physical properties. This is important in taking a sample for chemical analysis.

[1] *Loc. cit.*
[2] *Min. Sc. Press*, 1908, XCVI, 560.
[3] Austin, *loc. cit.*
[4] *Min. Res. U. S.*, 1883–84, p. 440.
[5] *Proc. Colo. Sc. Soc.*, 1891–93, IV, 304, and private communication, April, 1897.

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TABLE 66.—ANALYSES OF SLAGS OF SAME COMPOSITION BUT DIFFERENT CRYSTAL FORMS

	Globular radiated form	Apparently cubical form	Prismatic form	Liquid slag
SiO_2	36.50	36.60	37.1	37.20
Fe	20.30	20.00	20.4	22.10
Mn	2.10	2.20	2.0	1.80
Al_2O_3	7.80	7.40	7.2	N. d.
CaO	23.10	21.00	21.9	18.00
BaO	0.52	0.66	0.92	1.30
MgO	Trace	Trace	Trace	N. d.
Pb	N. d.	N. d.	N. d.	1.50
S	0.40	0.36	0.55	N. d.

Tenacity.—There is some variety in the tenacity of slags. Silicious slags are generally tougher than those in which the base prevails. The more crystalline a slag, the more brittle. A pot of slag may show brittleness in one part and toughness in another.

Specific Gravity.—The specific gravity of a slag is an important factor in its separation from lead and particularly from matte, which has a specific gravity of about 5. Iles[1] gives, as extreme figures, 3.3 and 4.16; as a common range for good slags, 3.4 and 3.6. The average specific gravity of 100 determinations of good slags gave him 3.691, and 3.6 is accepted as the highest specific gravity a good slag ought to have.

Diaphaneity.—Slags do not possess to any extent the property of transmitting light. Single crystals in exceptional cases are transparent; sometimes they are translucent, but generally opaque.

Color.—Lead slags are usually black from their high percentage of iron. Thin pieces sometimes appear greenish black; a large amount of iron will give a brownish hue. Lime produces a bluish or grayish tone.

Luster.—The luster of slag varies. It is occasionally metallic, but generally vitreous to resinous. Very often slags are dull.

Magnetism.—Some lead slags show magnetic properties caused by the presence of Fe_3O_4 or magnetic iron sulphide, Fe_8S_9. The Fe_3O_4 may be due to imperfect reduction of Fe_2O_3, or to the use of magnetite as a flux, it may be formed by the reaction: $4PbO+3Fe=Fe_3O_4+4Pb$.[2] In all probability its formation is governed by the SiO_2-content of the slag, a high percentage being unfavorable to its formation.[3] Neill,[4] in examining for their magnetic properties a large number of lead slags running from 28 to 34 per cent. FeO, found that they were not attracted by the magnet; but with higher percentages,

[1] *Loc. cit.*

[2] Berthier, "Traité des Essais par la voie sèche," Thomine, Paris, 1834, I, 385.

[3] Wheeler-Krejci, *Tr. A. I. M. E.*, 1913, XLVI, 562.

[4] *Tr. A. I. M. E.*, 1891, XX, 586.

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e.g., Eilers' slag, SiO_2 28, FeO 52, CaO 12, they were attracted; samples with 40 per cent. FeO were not attracted.

134. Fluxes.—Silica.—In Fig. 318 it was seen that the formation temperature of a ferro-calcic slag fell as the SiO_2-content rose. Hofman[1] has shown experimentally, Fig. 320, that this fall continues to the degree of a 2.5-silicate; experiences with the iron blast furnace and the glass pot point in the same direction. The fluidity of a slag decreases with the increase of SiO_2; silicious slags are more viscous than basic.

As most silver-lead ores are silicious, it is rarely necessary to add silicious ore to the charge. However, lead carbonates and many galena concentrates frequently contain an excess of base and require a silicious addition. In such cases silicious ores containing lead or silver or both will be used as flux, if available, instead of barren sandstone. In choosing a silicious material the amount of available SiO_2 will have to receive due consideration, §152.

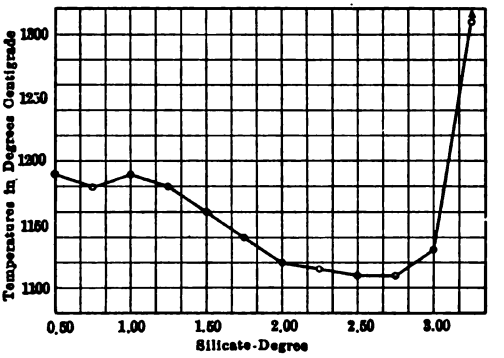

FIG. 320.—Effect of increase of SiO_2 upon formation temperature of a ferro-calcic silicate.

135. Iron.—In Fig. 318 it is seen that with ferrous silicates the formation temperatures rise with the iron contents. Silicious irony slags run like water.

An iron flux acts in three ways: It furnishes a base for the SiO_2 of the ore: $FeO.SiO_2+FeO=2FeO.SiO_2$. Being reduced to Fe by C or CO it acts as a precipitating agent with galena: $4PbS+2Fe_2O_3+3C=4Pb+4FeS+3CO_2$, or $2PbS+4FeO.SiO_2+C=2Pb+2FeS+2FeO.SiO_2+CO_2$; and $2PbS+Fe_2O_3+3CO=2Pb+2FeS+3CO_2$; none of the reactions are as complete as expressed by the formulæ. Further, FeO may liberate PbO from its combination with SiO_2 which later is reduced by C or CO to Pb, as indicated by $4FeO.SiO_2+2PbO.SiO_2+C=2Pb+4FeO.6SiO_2+CO_2$.

Slags rich in FeO have a high specific gravity and are dangerous, as they are likely to cause the formation of crusts. A slag high in FeO, however, is a necessity, if much Zn is contained in the ore, as FeO favors the slagging and solution of ZnO.

The purer the iron ore, the greater will be its fluxing power, as only that part of the FeO is available which is not required by its own SiO_2 (S or As). This SiO_2 not only limits the fluxing power, but also consumes limestone to form slag. With increase of slag-forming constituents the amount of space in the blast furnace for lead ore, of course, diminishes; the loss by scorification on the other hand increases, but the larger the amount of slag formed, the greater will be the quantity of ZnO it can carry away.

The common iron ores are hematite, Fe_2O_3, or limonite, $Fe_2O_3+xH_2O$;

[1] *Tr. A. I. M. E.*, 1899, XXIX, 701.

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sometimes siderite, $FeCO_3$. The use of magnetite, Fe_3O_4, as a flux has given rise to some discussion.[1] The fact is that magnetite is difficult of reduction; hence a furnace using it as flux must have strong reducing conditions. As these are not easy to control, there may be over- and under-reduction. It has been shown in §133 that a high-SiO_2 slag is unfavorable to the formation of Fe_3O_4; hence with magnetite as a flux a low-SiO_2 slag has to be avoided.

136. Manganese.—The fluxing property of MnO is similar to that of FeO, and MnO is usually figured in a charge as equivalent to FeO, the molecular weights being 55 and 56. Hofman[2] has shown, Fig. 321, that the formation temperature of a ferro-calcic slag is raised by substituting MnO for FeO. It is known from practical experience that MnO–FeO slags have a greater fluidity than FeO slags.

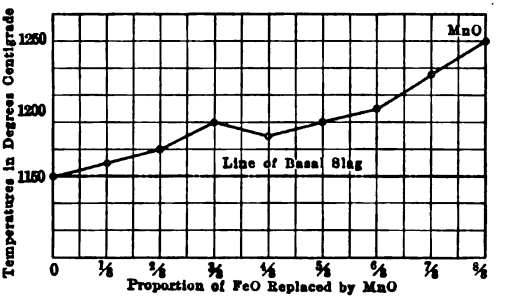

FIG. 321.—Effect of MnO upon formation temperature of a ferro-calcic silicate.

The series MnO–SiO_2 has been studied by Doerinkel[3] and the bi-silicate series of MnO and CaO by Ginsberg.[4]

The commonest manganese mineral is pyrolusite, MnO_2. Its oxidizing power has been said to be the cause that certain slags rich in MnO have so high a tenor in Ag. This seems to be contradicted by the fact that slags rich in MnO, Table 67, have been made by Church[5] that ran low in Ag and Pb, the lead bullion averaging 314 oz. to the ton.

The oxidizing power of oxides of manganese on blende in the reverberatory furnace, when sulphide copper ores are smelted, is mentioned by Pearce,[6]

TABLE 67.—ANALYSES OF SLAGS RICH IN MANGANESE AND POOR IN SILVER

SiO_2	FeO	MnO	CaO	MgO	Al_2O_3	Ag, oz. per ton	Pb
29.60	11.56	43.25	7.50	Trace	6.34	Trace	1.4
33.00	14.22	25.78	13.10	1.00	4.20	0.5	1.0

who obtained a slag of SiO_2 48 per cent., MnO 30 per cent., and ZnO 12.5 per cent., some Mn (3 per cent.) also entering the matte (50 to 60 per cent. Cu) as MnS. The oxidizing power for blende has been noticed in blast

[1] Colquhoun, *Min. Sc. Press*, 1905, XC, 69.
Neill, *loc. cit.*, 69, 133.
[2] *Tr. A. I. M. E.*, 1899, XXIX, 704.
[3] *Metallurgie*, 1910, VIII, 201.
[4] *Zt. anorg. Chem.*, 1908, LIX, 346.
[5] *Tr. A. I. M. E.*, 1886–87, XV, 612; *School Min. Quart.*, 1883–85, V, 322.
[6] *Tr. A. I. M. E.*, 1882–83, XI, 59.

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roasting in a Dwight-Lloyd sintering machine. Iles believes that in the blast furnace the amount of matte and speiss formed diminishes with the increase of Mn in the charge. Harbordt[1] gives it as his experience that the percentage of matte formed is not affected unless the Mn is present in considerable quantities. Furman[2] says that with from 10 to 12 per cent. MnO_2 in the charge the oxidizing power is noticeable. Another peculiarity of MnO is that it reduces the dissolving power of a slag for ZnO, MgO, BaS, and Met. As; some metallurgists hold that it increases the solubility of Met. S.

137. Lime.—The manner in which CaO replaces FeO in a slag can be expressed by $4FeO.2SiO_2 + 2PbS + 2CaO + 2C = 2Pb + 2FeS + (2CaO.SiO_2 + 2FeO.SiO_2) + 2CO$.

The earliest record of making a high-CaO slag in a lead blast furnace is that by Grüner[3] who states that at La Pise the furnaces were run on the slag, SiO_2 30, FeO 40, CaO 20 per cent. This slag was brought into prominence in the United States by Anton Eilers, and has become a stand-by (type E, Table 64). The first man to make independently a high-CaO slag was August Raht who in 1881[4] created the typical slag, G, Table 64, with SiO_2 35, FeO 27, CaO 28 per cent.

The effects that CaO has upon the formation temperatures of FeO-silicates have been shown in Fig. 318. The low formation temperatures of the general run of calcareous slags makes them more fluid at the prevailing blast furnace temperatures than the ferrous slags of the same silicate degree; hence CaO is said to increase the fluidity of a slag.

Schneider[5] found that with slags containing much CaO less matte is formed than when they are rich in FeO, and also that the matte is lower in Pb and higher in Ag. He explains it by saying that CaS is formed and then dissolved by the slag. Its presence in Leadville slags has been proved by Guyard,[6] in copper slags by Baikoff,[7] and in laboratory experiments by Schütz.[8] The solubility of Met. S-ide in slag is small,[9] as slag at the utmost contains from 2.5 to 3 per cent. S. The dissolving power increases with the temperature and basicity, and varies with the character of the bases. Jüptner[10] gives the S-content of a ferro-calcic bi-silicate as 0.06 per cent., and of a singulo-silicate as 0.62, 2.24 and 3.17 per cent.; he finds that an increase in the percentages of CaO, FeO, MnO, and ZnO has a tendency to raise the S-content. The use of CaO in slags is limited by foreign matter in the ore, and especially by the presence of Zn. In a general way it may be

[1] Private communication, July, 1891.
[2] *School Min. Quart.*, 1892–93, XIV, 315.
[3] *Ann. Min.*, 1868, XIII, 367.
[4] Eilers, *Tr. A. I. M. E.*, 1915, LII, 734.
[5] *Tr. A. I. M. E.*, 1882–83, XI, 58.
[6] *Loc. cit.*, p. 73.
[7] Hofman-Mostowitch, *Tr. A. I. M. E.*, 1910, XLI, 783.
[8] *Metallurgie*, 1907, IV, 697.
[9] Hofman, "General Metallurgy," 1918, p. 465.
[10] *J. Iron and Steel Inst.*, 1902, I, 327.

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said that with zincky ores the less CaO and the more FeO a slag contains the better will the furnace work. It does not seem advisable to go beyond 16 per cent. of CaO in a slag if from 9 to 12 per cent. of ZnO is present in the charge. For example, the quarter slag, C, Table 64, will readily hold 10 per cent. ZnO; the half slag, E, 7.5 per cent.; the three-quarters slag, F, 6.5 per cent.; the one-to-one slag, G, hardly any at all. If the CaO-content exceeds 28 per cent. (according to some metallurgists, 24 per cent.), ZnO will not enter the slag, but forms in part a mush which incrusts the furnace, and is in part volatilized. A high SiO_2-content appears to assist the ZnO to enter the matte; a low, the slag.

The use of CaO lowers the specific gravity of the slag and therefore favors the separation of matte and slag.

Limestone, $CaCO_3$, is the rock which is used to furnish the necessary CaO. Under atmospheric pressure it is dissociated into CaO and CO_2 at 910° C.[1] The purer the mineral, especially the lower the SiO_2 and Al_2O_3, the greater will be the CaO-content available as flux.

Burnt lime is rarely, if ever, used in the blast furnace. Bretherton[2] states that at the American Smelter, Leadville, Colo., he used for 8 years burnt lime instead of limestone, with the result not only that the furnaces put through more ore with the same labor and fuel than with limestone, but that cleaner slags were produced. Some of the leading Colorado lead smelters experimented systematically for months with burnt lime and, finding no metallurgical or economic benefit, returned to the use of limestone.

Burnt lime was used many years ago in the iron blast furnace, and is still so used in some instances in England.

The supposed advantages were that more ore could be put through per unit of fuel in a given time (as 56 CaO charged are equivalent to 100 $CaCO_3$); that the heat consumed by driving off the 44 CO_2 in the blast furnace was saved; and that the partial reduction of the 44 CO_2 by means of C did not occur. Grüner[3] calculated a saving of 10 per cent. of fuel by the use of burnt lime; Bell[4] made it only 3.4 per cent. The reason why burnt lime in the iron blast furnace has been given up is that it absorbs CO_2 and H_2O-vapor from the furnace gases, a heat up to redness[5] assisting the absorption of CO_2, and CaO_2 parting even less readily with the combined H_2O than with CO_2. Though the absorption generates the same amount of heat that the expulsion consumes, it occurs in the upper parts of the furnace, warming the gases that are passing out; meanwhile the lower part is being cooled by the expulsion, and this has to be made up by an extra amount of fuel. If lime is to be used, the lime must be burned in large pieces and the burning must be very complete and even carried to sintering, if possible (*i.e.*, with impure limestone), so as to diminish the absorption of CO_2 and H_2O. The appa-

[1] Zavrieff, *Compt. rend.*, 1907, CXLIV, 428.
[2] *Eng. Min. J.*, 1893, LV, 196.
[3] *Ann. Min.*, 1871, XX, 325; and *Dingler's Polytech. J.*, 1872, CCIV, 309.
[4] "Chemical Phenomena of Iron Smelting," London, 1872, p. 138.
[5] Bell, "Manufacture of Iron and Steel," London, 1884, p. 60.

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ratus for burning lime must be easy to regulate and manage, so as to supply just the amount of lime required by the blast furnace, as storage and transportation for any distance are out of the question. The fuel charged into the blast furnace must be dry, and the ore and iron flux not only dry, but free from CO_2. Finally, the height of the furnace may have to be increased to prevent the furnace gases from passing off at too high a temperature. The many difficulties in the way of using burnt lime have deterred the iron smelter from it, and special economic conditions must prevail to make it profitable to the lead smelter, as for example, when on account of high freight charges burnt lime can be delivered at the smeltery at the same cost as limestone. The use of burnt lime in the iron blast furnace was again discussed by iron metallurgists.[1]

138. Magnesia.—The effect of MgO upon a lead slag is to raise the formation temperature. This is clearly shown in Fig. 322. Here the formation temperature, 1,150° C., of the basal slag, SiO_2 32.10, FeO 35.90, CaO 32.00 per cent., is seen to rise as soon as MgO is introduced to replace CaO in equivalent amounts. Beside raising the formation temperature, MgO makes the slag pasty and streaky. A slag may contain as much as 5 per cent. MgO and still run well from the overflow of a fore-hearth; if it contains more than 5 per cent. there is likely to be trouble.[2] If there is little matte formed, and a small overflow-pot is used instead of a large fore-hearth, the content of MgO may go as high as 8 per cent., but even then the slag will chill very quickly when exposed to the cooling effect of the air (Federal, Ill.).

This undesirable property of making a slag sticky is especially noticeable if the slag contains some ZnO; the two bases appear to intensify their properties of being difficult to slag. This is seen in Fig. 322. In a slag containing 8 per cent. ZnO and from 2 to 3 per cent. BaO, the presence of from 2 to 3 per cent. MgO shows a decidedly bad effect, and 5 per cent. causes trouble, the slag showing a tendency to freeze. Silicate of magnesia has a higher specific heat than silicate of lime.

In calculating a charge, MgO is figured as replacing an equivalent amount of CaO, viz., $CaO = MgO \times 1.4$.

139. Alumina.[3]—From the composition of the typical lead slags it will be seen that the place Al_2O_3 occupies in lead smelting is generally a subordinate one. When it is present in large quantities, it becomes a question whether it acts as an acid or a base. It is known in a general way that with a high percentage of SiO_2, the Al_2O_3 acts as a base; with a low percentage it acts as an acid. Iles[4] gives this as his experience in lead smelting. Austin[5] believes that with slags up

[1] Bell, *J. Iron and Steel Inst.*, 1893, II, p. 274; 1894, II, 38.
Howden, *op. cit.*, 1894, I, 85.
Cochrane, *op. cit.*, 1894, II, 62.
Howe, *op. cit.*, 1894, II, 87.
Kosmann, *Stahl und Eisen*, 1891, XI, 311.

[2] Private communication by W. Allen Smith, August, 1917.

[3] Henrich, *Tr. A. I. M. E.*, 1916, LVI, 621, 943.

[4] *Min. Res. U. S.*, 1883–84, 433.

[5] *Eng. Min. J.*, 1904, LXXVIII, 253.

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to 36 per cent. SiO_2, the Al_2O_3 acts neither as a base nor as an acid, but is simply dissolved in the slag acting as a stiffening ingredient. A similar opinion is held, without limiting the SiO_2-content, by Dwight and Mathewson[1] for lead and copper blast furnaces; Boggs[2] found that in a ferruginous copper blast furnace slag, he could neglect Al_2O_3 in his charge calculations as long as its amount

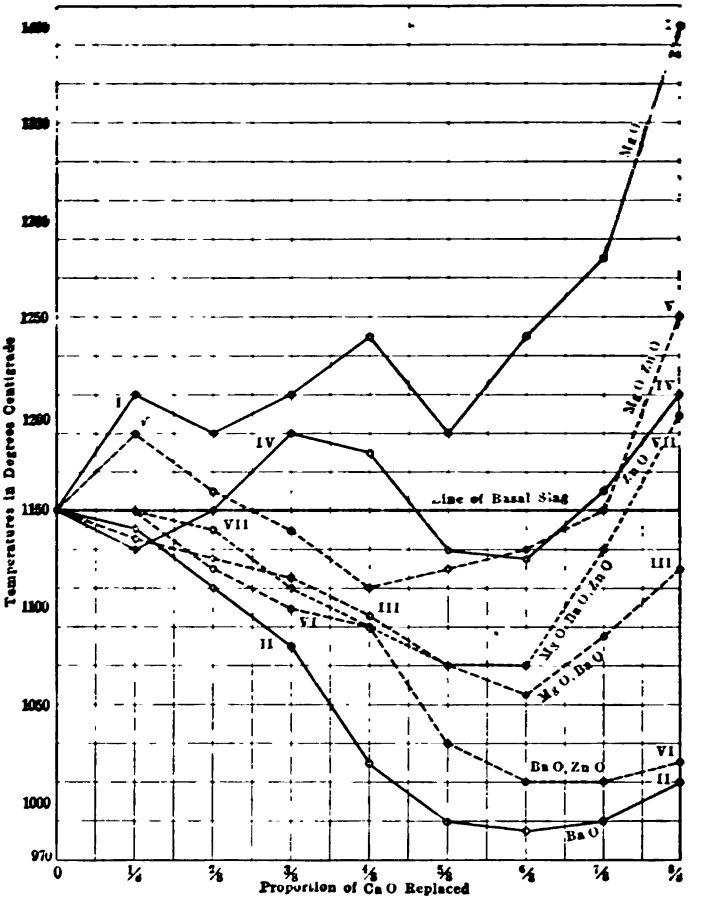

FIG. 322.—Effects of MgO, BaO, and ZnO upon formation temperature of a ferro-calcic silicate.

did not exceed 10 per cent. Mathesius[3] and Johnson[4] state from their experience with iron blast furnaces that Al_2O_2 is chemically indifferent in the formation of slag; this does not agree with the researches of Rankin and Wright of the ternary system CaO–Al_2O_3–SiO_2.[5]

[1] *Tr. A. I. M. E.*, 1916, LVI, 627, 942.
[2] *Loc. cit.*
[3] *Stahl u. Eisen*, 1908, XXVIII, 1121.
[4] *Tr. A. I. M. E.*, 1912, XLIV, 123; 1916, LVI, 941.
[5] *Am. J. Science*, 1915, CLXXXIX, 1.

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Hahn[1] thinks that Al_2O_3 always acts as a base, and says that an increase of Al_2O_3 requires also an increase of SiO_2, or, what would be the same, a decrease in the bases. Schneider[2] found that as a general rule an increase of Al_2O_3 called for an increase in the proportion of CaO. This may mean a decrease in SiO_2, the Al_2O_3 acting as an acid.

Howe,[3] summing up the statements of Hahn and Schneider, suggests that the part played by Al_2O_3 may depend upon the proportion of the other two fluxes, CaO and FeO, and that in calcareous singulo-silicates, low in FeO, Al_2O_3 may act as an acid, and in ferruginous slags, low in CaO, as a base. The idea seems to be confirmed by the experience of Peters[4] in smelting Mount Lincoln[5] ores in Colorado. Henrich[6] believes that Al_2O_3 always acts as an acid; he

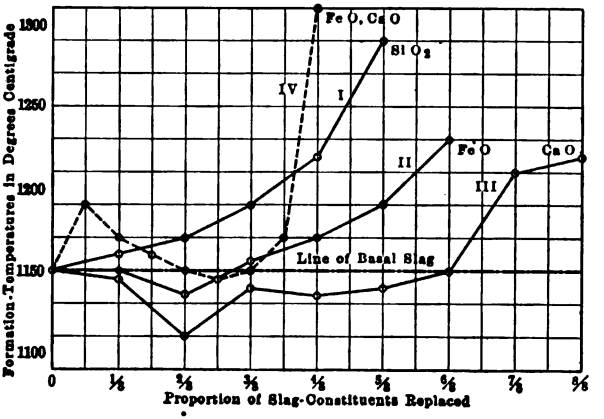

FIG. 323.—Effect of Al_2O_3 upon formation temperature of a ferro-calcic silicate.

proposes two types of silicate aluminates: mono-silicate-aluminate ($2Al_2O_3.3SiO_2$) : (8 Metal Oxide+4 Earth Oxide); $O_{acid} : O_{base} = 1 : 1$. Bi-silicate-aluminate ($2Al_2O_3.9SiO_2$) : (8 Metal+4 Earth Oxide); $O_{acid} : O_{base} = 2 : 1$.

The experimental evidence of Hofman,[7] given in Fig. 323, shows that replacement of SiO_2 by Al_2O_4 raises the formation temperature of 1,150° C. of the basal slag (SiO_2 32.10, FeO 35.90, CaO 32 per cent.), and that replacement of CaO lowers it to a certain extent. If the two replacements are combined, it ought to be possible to maintain the original formation temperature of 1,150° C. Practical experience in smelting lead-bearing brick bats and other aluminous material of a refinery has corroborated this. If, *e.g.*, with a slag, SiO_2 36, FeO 33, CaO 15 per cent., an aluminous material is to be smelted, parts of both

[1] *Min. Res. U. S.*, 1882, p. 328.
[2] *Tr. A. I. M. E.*, 1882–83, XI, 57.
[3] *Eng. Min. J.*, 1883, XXXVI, 306.
[4] *Ibid.*, p. 322.
[5] *Tr. A. I. M. E.*, 1882–83, II, 310.
[6] *Eng. Min. J.*, 1886, XLII, 40, 75, 128; *Bull. A. I. M. E.*, November, 1916, p. 2081; February, 1917, p. 2081; June, 1917, p. 989; September, 1917, p. 1473 (Johnson).
[7] *Tr. A. I. M. E.*, 1899, XXIX, 710.

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SiO_2 and CaO are best replaced by Al_2O_3, and the FeO left intact. The slag produced would have the following composition: SiO_2 30, FeO 33, CaO 12+1.6 (equivalent to $3Al_2O_3$), Al_2O_3 6(replacing $6SiO_2$). The presence of Al_2O_3 has a tendency to make the slags glassy. If it is not found necessary to reduce the SiO_2-content of the slag, the Al_2O_3 is figured into a charge as replacing CaO, that is, $103Al_2O_3 = 56CaO$, or roughly $2Al_2O_3$ replace 1CaO.

140. Fluorspar.—This is of less importance at present than used to be the case. It is known that with $BaSO_4$ and $CaSO_4$ it forms readily fusible mixtures, that it assists in the fluxing ZnS and ZnO, but the manner of its action has not yet been explained. The only research along this line is that of Karandéef[1] who found that CaF_2, melting at 1,378° C., formed with $CaSiO_3$, melting at 1,501° C., an eutectic with 38.2 per cent. CaF_2 freezing at 1130° C.

Its chemical effect in volatilizing Si as SiF_4 need not be taken into consideration. Foehr[2] claims that the presence of from 1 to 5 per cent. CaF_2 in the roasting of ores in the reverberatory furnace saves fuel, and that adding it to the charge in refining lead prevents shots of lead from being retained by the litharge; but gives no proofs.

If fluorspar is used in a smelting charge, one has to remember that its Ca is not available as a base for fluxing SiO_2, but remains in combination with F.[3] Ores from northwestern Mexico often contain CaF_2; the percentage of F has to be known before a correct calculation of the amount of CaO needed can be carried out.

141. Slag.—There are four reasons for the use of slag in blast furnace charges: (1) It may contain too much Pb or Ag to be thrown away. (2) It makes the charge less dense. (3) It helps the actual smelting process because the slag, having been already melted, will remelt easily and promote the smelting of the ore itself; and (4), if it be more acid or basic than the slag that is being formed by the smelting mixture, it will act as an acid or basic flux.

With a furnace running in a normal way, some rich (foul) slag is always produced. This is especially the case when the last slag in the furnace is being tapped, and the blast passing through the tap-hole blows out valuable parts, which enrich the slag in the pot (blow-pot). Then again, when much matte comes out with the slag, it is likely not to settle out perfectly.

It used to be the custom to add waste, especially foul, slag to all blast furnace charges. With fine ores the addition amounted to as much as 25 per cent.; with coarse ores it reached 10 and 15 per cent. With fine ores the use of waste slag is necessary in order to keep open the charge so that the ascending gases pass evenly through it and do not form blow-holes. With coarse ores, as first proved by Wm. Wraith at Tooele, Utah,[4] this addition is not only un-

[1] *Zt. anorg. Chem.*, 1910, LXVIII, 188.
[2] *Eng. Min. J.*, 1890, XLIX, 706, 735.
[3] Austin, *Eng. Min. J.*, 1905, LXXX, 865, 1222.
Kneeland, *ibid*, 1030, 1212.
Dwight, *Bull. A. I. M. E.*, April, 1917, p. 469.
[4] Anon, *Eng. Min. J.*, 1916, CII, 1100.

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necessary, but distinctly harmful. Space is taken up by the slag instead of by ore-charge; fuel is required for smelting it; the return through values recovered is too small to pay for the cost; the ZnO of the slag is returned to the blast furnace and interferes with clean smelting. In our days, a large part of the charge is made up of blast roasted material so that a loosening of the charge through waste slag is unnecessary; attention is given to a thorough settling of matte in one or two fore-hearths when the metal retained by the slag is present in such a form that very little is recovered by resmelting.

Slag more basic than the normal slag comes into play where, for example, matte is being concentrated in a reverberatory furnace, and the resulting slag contains much iron that is available for the blast furnace.

In smelting the by-products of desilverizing works in the blast furnace without ore, the amount of slag added goes up as high as 50 per cent.

142. Influence of Foreign Matter, Gypsum.—The mineral gypsum, $CaSO_4 + 2H_2O$, does not occur frequently with lead ores; nor is it commonly used as a flux to furnish CaO for the slag; it enters, however, the blast furnace charge with blast roasted material.

Gypsum begins to lose its combined water at 149° C., and is dehydrated[1] at 255°. The $CaSO_4$ formed[2] remains unchanged up to 1200°; dissociation into CaO, SO_2, and O begins at 1200° and increases at an accelerating speed with rise of temperature; at 1400° the partly decomposed sulphate fuses. In presence of SiO_2, the dehydrated $CaSO_4$ begins to be converted into $CaSiO_x$ at 1000° C., the formation of silicate is complete at 1200°. Ferric oxide, Fe_2O_3, begins to decompose $CaSO_4$ at 1100°, the decomposition is complete at 1250° with the formation of a ruby-colored eutectic.[3] Heating with the reducing agents CO and C[4] reduces $CaSO_4$ to CaS without loss of S. The reduction by CO begins at 700° C. and is finished at 900°; that by C in a neutral atmosphere begins also at 700° C., but is finished only at 1000°. Roasting CaS in pure dry air gives a product of $CaSO_4$ 76 and CaO 24 per cent. The process is accompanied by a loss in S caused by the interaction of $3CaSO_4$ and CaS, which begins at 800° C. and is intensified with a rise of temperature; the reaction, however, is not complete, owing to the CaO formed, which forms a barrier between the active agents. If $CaSO_4$ is to act as a sulphurizing agent,[5] the ore, gypsum, and carbon must be intimately mixed, the mixture being usually briquetted; the formation of matte must take place in the upper part of the furnace where the charge is still solid, and the gases in the furnace must be strongly reducing, that is, CO must prevail largely over CO_2.

143. Barite.—This mineral occurs frequently with silver-lead ores. Pure $BaSO_4$[6] upon heating begins to give off SO_3 at 1510° C., fuses at 1580°, and is

[1] Hofman-Wanjukow, *Tr. A. I. M. E.*, 1912, XLIII, 570.

[2] Hofman-Mostowitsch, *op. cit.*, 1909, XXXIX, 624; XL, 807.

[3] Kohlmeyer-Hilpert, *Metallurgie*, 1910, VII, 194, 225.
Sosman-Mervin, *Wash. Acad. Sc.*, 1916, VI, 532.

[4] Hofman-Mostowitsch, *Tr. A. I. M. E.*, 1910, L, 763.

[5] Hofman-Mostowitsch, *loc. cit.*, 783.

[6] Mostowitsch, *Metallurgie*, 1909, VI, 450; *Eng. Min. J.*, 1909, LXXXVIII, 601.

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thereby further decomposed, but not wholly, as the fused mass is a mixture of $BaSO_4$, and BaO. The presence of small amounts of impurities, *e.g.*, a trace of Fe, causes the dissociation to begin at a lower temperature. In the presence of SiO_2, the decomposition begins at 1000° C. with the formation of silicate. Sub- and singulo-silicates begin to sinter at 1400°; sesqui-, bi- and tri-silicate mixtures sinter at 1350°, and fuse at 1400°. The decomposition by Fe_2O_3 begins at 1100° C., and is not so energetic as that by SiO_2. The action of Fe_2O_3 increases with the amount used and with the temperature; thus $1BaSO_4+2Fe_2O_3$ fuses at 1350° to a liquid consisting of BaO and Fe_2O_3; $1BaSO_4+1FeO_3$ and $1BaO+3Fe_2O_3$ require 1400° for liquefaction, and even then the decomposition of $BaSO_4$ is not complete.

Decompositions of $BaSO_4$ by SiO_2 in the presence of Fe and FeS have been formulated by Balling:[1] $2BaSO_4+8Fe+4SiO_2=(BaS.FeS)+7FeOBaO.4SiO_2$, and by Schweder:[2] $BaSO_4+Fe+2SiO_2=BaSiO_3+FeSiO_3+SO_2$, and $3BaSO_4+FeS+4SiO_2=3BaSiO_3+FeSiO_3+4SO_2$. The first equation explains the presence of BaS in a matte in the absence of C; equations 2 and 3 presuppose that SO_2 and O have an oxidizing effect upon Fe and FeS. Ordinarily very little BaS is found in a matte; there are, however, cases on record[3] in which matte contains from 4 to 24 per cent. Ba.

Reduction[4] of $BaSO_4$ to BaS by C begins at 600° C., and is complete at 800°; the reduction by CO begins at 650°, reaches 98 per cent. at 800°, 99.1 per cent. at 900°, and is complete at 1050°. The BaS formed is stable at 1000°, but gives off some S at 1200°. The investigations of Wells[5] show that with H the reduction begins at 550° C., that between 600 and 700° much H_2S is evolved, that at 800° the issuing gas contains very little H_2S, and that in the product the ratio of S to Ba is less than 1 : 1. He found that the reduction with CO is complete above 900°, and that some S is volatilized in the process; also that the reduction with C is completed only between 1000 and 1100°.

The great fluxing power of BaO is shown in Fig. 322. The formation temperature of the basal slag (SiO_2 30.10, FeO 35.90, CaO 32.00 per cent.) is seen to be greatly and evenly lowered through the replacing of CaO by BaO until six-eighths of the CaO has been replaced, when the temperature rises again. The slags formed are very fluid. The refractory natures of both MgO and ZnO are readily overcome by Ba, as not only are their formation temperatures lowered, but their irregular behaviors corrected. Thus BaO would be a very desirable flux, were it not for its high specific gravity which makes the slags heavy and thus hinders the desired separation of matte and slag.

In computing a charge, BaO is figured as replacing CaO; $BaO=CaO\times0.38$. From what has been said, it is clear that all the S contained in $BaSO_4$ can not

[1] "Compendium der Metallurgischen Chemie," Strauss, Bonn, 1882, p. 89.
[2] *Berg. Hüttenm. Z.*, 1886, XLV, 547.
[3] Jossa-Kurnakoff, *Berg. Hüttenm. Z.*, 1879, XXXVIII, 38; *Iron*, 1879, XIII, 387.
[4] Mostowitsch, *loc. cit.*
[5] Bureau of Mines, *Bull.* 113, 1917; *J. Ind. Eng. Chem.*, 1916, VIII, 770.

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be figured into the charge; if 10 per cent. of it is assumed to enter the matte, ample justice will have been done.

144. Blende.—This mineral ZnS, which is found with most lead ores, causes no end of difficulty in the blast furnaces. It is practically infusible; it is decomposed[1] by iron oxides and silicates, the resulting ZnO entering the slag to a greater or less degree; metallic Fe liberates Zn. Most of the ZnS in a blast furnace charge remains undecomposed, and enters matte as well as slag; it makes both less fusible, obstructs separation of matte and slag, and therefore causes matte to remain entangled in slag.[2] If present to any extent, it forms a mushy matte which floats upon the normal matte and is difficult to tap; on analysis such a mush gave Easter[3] Insol. 6.6, Fe 19, CaO 3.4, Zn 24.3, S 21, Pb 10, Cu 4.6 per cent.

The addition of chalcopyrite[4] mends matters; but, as seen from Easter's analysis, its power is limited.

If blende is present to a considerable extent in the ore, this must be roasted before it is smelted; blast roasting leaves a large part of the ZnS unchanged, as the operation is too quick for satisfactory oxidation. The roasting process is of course connected with loss, especially that of Ag. Simmonet[5] claims that with an addition of coarsely crushed limestone to the roasting-charge, most of the Pb and Ag pass into the more or less altered limestone, which then can be separated by screening from the finely divided zinc ore. The writer's experiments with this mode of operating have been anything but successful. Iles[6] avoids the use of metallic Fe for the decomposition of ZnS, as the Zn-vapor liberated favors the formation of wall-accretions.

The progress made in ore-dressing has greatly reduced the earlier difficulties encountered by the presence of blende, as this is generally removed before the ore is smelted. There exist, however, many deposits in which galena and blende are so intimately associated that a satisfactory mechanical separation has not been accomplished, at least so far. Two processes which may solve this difficult problem are on trial at present. One is electric smelting, which aims to recover lead, zinc, and matte, and to slag the remaining constituents of the ore; the other, roasting followed by lixiviation with H_2SO_4 and electro-deposition of Zn, using insoluble anodes, the lead-bearing residue going to the blast furnace. Electric smelting has not been very successful. Many smelteries have erected leaching and depositing plants which work satisfactorily; the cost of the spelter produced is high, but the cost has to be borne by the ore, which otherwise could not be treated at all. The treatment of zinc-lead sulphides is a separate chapter which is better discussed under zinc than lead.

[1] Plattner, *Berg. Hüttenm. Z.*, 1854, XIII, 81.
[2] Iles, *School Min. Quart.*, 1898, XIX, 197.
[3] *Tr. A. I. M. E.*, 1915, LII, 725.
[4] Douglas, see *Bull. A. I. M. E.*, April, 1917, p. 765.
Hahn, *Min. Res. U. S.*, 1882, p. 343.
Keller, *Tr. A. I. M. E.*, 1892–93, XXI, 71
[5] *Ann. Min.*, 1870, XVII, 27.
[6] *School Min. Quart.*, 1896–97, XVIII, 18.

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The curves of Fig. 322 corroborate the practical experience that ZnO and MgO intensify the undesirable properties of each other; they also show that BaO overcomes them readily.

There is much speculation in regard to the form in which ZnO is present in a blast furnace slag. From some slags a large percentage of ZnO has been removed by leaching with NH_4O or dilute H_2SO_4, which points to ZnO. The insoluble part is considered to be either $ZnSiO_3$[1] or zinckiferous magnetite[2] or iron-bearing zinc spinel, $ZnAl_2O_4$[3] which may separate from zinc-iron slags with, say, less than 10 per cent. Al_2O_3.

145. Zinc Oxide.—This occurs in combination with CO_2, SiO_2, and H_2O in carbonate lead ores, or is formed in roasting blende-bearing sulphide lead ores. It is infusible in carbon-heated furnaces, but is reduced by C to Zn-vapor at temperatures ranging from 1007 to 1087° C. according to the physical nature of the oxide and the C,[4] and the Zn-vapor is readily oxidized by CO_2, O, and H_2O-vapor.

It is difficult to slag ZnO. This is seen in Fig. 322, in which replacement of CaO by ZnO in the basal slag shows great irregularity in the curve. This is due probably to the form in which the ZnO is present in the slag, as it may be simply held in igneous solution, or $ZnSiO_3$ may have been formed[5] and dissolved in the slag, or it may have combined with it in some unexplained manner. The slags made at Point Pirie, N. S. W., and Chillagoe, Queensland, show compositions which at first sight differ entirely from those that are common elsewhere, as seen in Table 67. If, however, the ZnO and Al_2O_3 are assumed to form zinc spinel, $ZnAl_2O_4$, and the slag is recalculated on that basis, it is seen that the contents of SiO_2, FeO, and CaO show nothing abnormal. The following analysis of a slag, recently made in Colorado, gives evidence of Zn being present in the insoluble residue, obtained in the usual way by boiling the chilled sample in HCl. The ordinary slag determination gave: Insol. 29.4 (SiO_2 26.1), Fe 28.7, Mn 0.7, CaO 13.4, Zn 8.0, Al_2O_3 7.3, Pb 0.9, Cu 0.41. The insoluble residue was fused with alkali, taken up with acid, dehydrated, the SiO_2 expelled with HF, and the residue analyzed, giving: Fe 0.58, CaO 0.07, Zn 0.6, Al_2O_3 1.28. It is held that with a zinc-bearing slag which contains over 6 per cent. Al_2O_3, there is a tendency to form zinc spinel.

It has been stated in §137 that if ZnO is to enter the slag, the percentage of FeO ought to be high, and that of SiO_2 and CaO low. Table 67 gives compositions of slags which have been run more or less successfully. The furnaces of

[1] Iles, *School Min. Quart.*, 1898, XIX, 197.
Hutchings, *Eng. Min. J.*, 1903, LXXVI, 959.

[2] Hutchings, *loc. cit.*
Vogt, see below.

[3] Stelzner-Schulze, *Berg. Hüttenm. Z.*, 1881, XL, 145, 150.
Vogt, J. H. L., "Die Silicatschmelzlösung," Dybwad, Christiania, 1903, 78, and "Mineralbildung in Schmelzmassen," Cammermeyer, Christiania, 1892, p. 199.

[4] Johnston, *Tr. A. I. M. E.*, 1907, XXXVIII, 656; 1913, XLVII, 219.

[5] Stein, *Zt. anorg. Chem.*, 1907, LV, 179, states that $ZnSiO_3$ melts at 1479° C., $ZnSiO_4$ at 1880°, temperatures measured with a Wanner pyrometer.

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the Sophien Works and of Thomasville had bottom-tapped crucibles; the Colorado and North Carolina slags run high in CaO; they perhaps show what may be done, rather than what should be done.

TABLE 67.—HIGH-ZINC LEAD BLAST FURNACE SLAGS

	Port Pirie,[1] N. S. W.		Chillagoe,[2] Queensland		Colorado[3]		Harz Mts.,[4] Prussia (*a*)		Thomasville,[5] N. C. (*a*)					Carlisle,[6] N. M.	
SiO_2	25.0	28.0	26.9	29.3	34	35	16.90	19.21	24.5	26.0	27.0	27.9	30.0	31 96	38.44
FeO	33.0	37.0	26.2	28.5	34	38	35.05	39.85	29.4	33.4	31.5	33.9	29.0	31.82	38.27
MnO	6.0	6.7	7.9	8.60	...	...	.....	.....	....	....	....	....	....	.....	.....
CaO	12.0	13.5	16.2	17.65	17	13	6.05	6.88	24.5	14.4	19.0	14.6	14.0	6.80	8.18
ZnO	13.0	9.2	12.5	9.70	7	12	19.64	16.61	14.5	19.8	15.0	16.6	15.5	13.84(*b*)	8.25
Al_2O_3	6.0	.....	4.5	.....	...	...	6.31	.....	....	....	....	....	....	8.77	.....
S	3.0	3.4	2.9	3.20	...	...	.....	.....	....	....	....	....	....	.....	.....
Pb	1.5	1.7	1.73	1.88	...	...	.....	.....	....	....	....	....	....	.....	.....
Cu	.....	.....	.....	.....	...	...	.....	.....	....	....	....	....	....	0.24 (PbO)	0.29 (PbO)
$BaSO_4$	.....	.....	.....	.....	...	...	10.24	11.64	....	.. .	....	....	....	Trace	Trace
Total	99.5	99.5	98.83	98.83	92	98	94.19	94.19	92.9	93.6	92.5	93.0	88.5	93.43	93.43
Pounds spinel in 100 lb. slag	12.2		8.1		.....		11.34		.....					15.75	

(*a*) Furnace crucible-tapped. (*b*) In addition, ZnS, 6.71.

[1] Delprat, *Tr. Austral. Inst. Min. Eng.*, 1907, XIII; *Eng. Min. J.*, 1907, LXXXIII, 317, 517.

[2] Poole, Sidney University, *Eng'g. Soc.*, Nov. 11, 1898.
Hofman, *Min. Ind.*, 1909, XVIII, 484.

[3] Editor, *Eng. Min. J.*, 1896, LXII, 194.

[4] Bräuning, *Zt. Berg. Hütt. u. Sal. W. i. Pr.*, 1877, XXV, 148.

[5] Canby, *Eng. Min. J.*, 1896, LXII, 292.

[6] Weinberg, *Eng. Min. J.*, 1896, LXII, 580.

As regards the behavior of ZnO in the blast furnace, it may be said that if it is reduced to Zn in the lower part of the furnace by C, or to a slight extent by Fe, it is almost instantaneously volatilized. The vapor ascending carries with it Pb and Ag, and, being oxidized higher up by CO_2 or H_2O-vapor or O from readily reducible oxide, carries off metal as dust, and forms oxide accretions on the sides of the furnace which begin about 7 ft. above the tuyères and increase in thickness to the charging floor. It is claimed[1] that a high-pressure blast counteracts this tendency. If this is the case, it cannot but be accompanied by increased losses in Pb and Ag. If ZnO, not reduced to Zn in its downward course with the charge, comes in contact with PbS or $PbSO_4$ in the presence of C, it may be converted into ZnS; in the presence of Fe it will remain unchanged.

Zinc silicate is completely reduced by C at a white heat; it is not known at what temperature the reduction begins.

If ZnO or $ZnSiO_x$ is to be slagged or dissolved, it may not be reduced to Zn, hence the smelting has to be done quickly and at a low temperature. This requires a slag not high in SiO_2 and with a preponderance of FeO. If, however, a slag high in CaO has to be made, it has become the practice to figure one-half of the ZnO as replacing an equivalent amount of CaO in the slag; CaO = ZnO×0.7. If this is done, it will be found that the total of SiO_2, FeO and CaO will have to be made larger than the usual 90 per cent. The amount of CaO a slag may contain and still work satisfactorily has been given in §137;

[1] Kloz, *Eng. Min. J.*, 1907, LXIII, 3581.

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it is further governed by the percentage of matte that is formed, the higher the matte-fall the lower the CaO-content of the slag.

Processes for the recovery of Zn from slags in the form of ZnO have been put into operation by Pape-Witter-Babe[1] and Divine.[2]

146. Pyrite.—The mineral FeS_2 is present in most lead deposits. Heated to 700° C. with exclusion of air, it loses one atom of S, leaving a residue which is attracted by the magnet.[3] The atom of S has to be provided usually with Fe to form FeS. There are on record cases in smelting anglesite ore in which the S passed off as SO_2 and comparatively little matte was formed.

Henrich[4] gives his successful experience in smelting carbonate ores at Benson, Ariz., consisting of galena (15 per cent.) and anglesite (75 per cent.), with silver-bearing pyrite. He obtained very little matte (20 lb. from 13 to 14 tons of ore), but considerable sulphur dioxide. This he explains as having been caused by the following reactions: $2FeS_2+5PbSO_4+SiO_2=5Pb+Fe_2SiO_4+9SO_2$; $PbSO_4+PbS=2Pb+2SO_2$. He says that the furnace ran rapidly and became very hot, so that the fuel (coke) had to be cut down from 12.5 to 11 per cent., the pressure of the blast being 1⅜ in. mercury.

147. Chalcopyrite.—In the blast furnace the aim is always to carry the Cu into the matte, which it enters as Cu_2S. Copper, having a greater affinity than any other metal for S, will generally take up all the S in the charge to form Cu_2S, and what is left is then available for Fe, Pb, etc. If a charge does contain Cu and not enough S to form Cu_2S, some Cu will be reduced to metal and be alloyed with the Pb. The alloy hardens, sinks to the bottom, and closes up the passage of the lead-well. Such an alloy contained Cu 35.1 per cent., Pb 47 per cent., and Ag 170 oz. per ton (Iles). There is one case where even with sufficient S to form Cu_2S the Cu combines with the Pb. This is when matte is concentrated in the blast furnace with a highly ferruginous slag. The affinities of Cu and Fe for S and SiO_2 seem to become disturbed.

If the slag be too basic, the Fe takes up some S and goes into the matte instead of separating and forming a crust; thus some S belonging to the Cu may be taken away, and this alloys with Pb. Another way of explaining the fact would be that reactions between sulphides and oxides of copper take place similar to those of the reverberatory furnace, and the resulting metallic Cu becomes alloyed with reduced Pb. Whatever may be the theory, the fact remains that any excess of Fe has to be avoided in the slag if the Cu is to be concentrated in the matte and not partly driven into the Pb. This is likely to occur when the matte contains about 12 per cent. Cu, and decidedly so with the increase of Cu.

[1] *Glück auf*, 1910, XLVI, 237; *Eng. Min. J.*, 1910, LXXXIX, 819; *Min. Ind.*, 1910, XIX, 453, 692; 1911, XX, 475.

[2] Pulsifer, *Met. Chem. Eng.*, 1915, XIII, 783.

[3] Valentine, *Tr. A. I. M. E.*, 1889–90, XVIII, 78.
Geodel, *J. Gasbeleuchtung*, 1905, XLVIII, 400.
Friedrich, *Stahl u. Eisen*, 1911, XXXI, 2040.
Barth, *Metallurgie*, 1912, IX, 204.

[4] *Eng. Min. J.*, 1883, XXXVI, 179.

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148. Antimony.—Antimony occurs either as Sb_2S_3 or an oxide. The Sb_2S_3 behaves on the whole like PbS, but is much more volatile. If decomposed by Fe the resulting Sb is more likely to combine with Pb than it is to form speiss with any excess of Fe that may be present. It may also be volatilized. The oxide is generally present as an antimoniate of lead or iron, and this, being reduced to an antimonide, combines with lead or speiss, if any is made, or with matte. The two main injurious effects of antimony, therefore, are that it causes loss by volatilization and impairs the character of the lead. Antimonial speiss is rare, and in making up an ore-charge no account need be taken of the small quantity of iron likely to be consumed by the antimony. It forms occasionally in smelting antimony skimmings (see §262).

Two difficulties have to be contended with in treating the antimonial by-products of refining works in the blast furnace. If the slag contains but little Fe, much antimony and lead are volatilized; if rich in Fe some speiss is likely to form, which either separates, causing the loss of the antimony, or becomes mixed with the slag, making it rich. A ferruginous slag is generally preferred to one that is calcareous.

149. Arsenic.—Arsenic occurs very frequently in argentiferous lead ores and must not be neglected in computing a charge. It causes loss by volatilization and combines with the Pb, but not to such an extent as Sb, as it has a great affinity for Fe and is likely to form a speiss. If it is necessary to consider the formation of speiss in making up a charge, it is advisable to figure 5 atoms As : 1 atom Fe, as this mixture is moderately fusible and pretty fluid when melted, does not readily form hearth accretions, and retains only very few shots of lead. It shows large cleavage planes similar to spiegel iron when broken. A speiss containing more or less Fe than called for by the supposed formula Fe_5As is not as easily fused, nor as fluid when melted; it is therefore more likely to form accretions and to hold lead in suspension.

In making up a charge, if the ore does not contain too much arsenic, it is possible to avoid producing any speiss at all by assisting part of the arsenic to pass off with the gases and the rest to combine with the lead and enter the matte. This can be done by making the charge open, by cutting down the fuel and thereby diminishing the reducing action, and by changing the proportions of FeO and CaO in the slag. It was once held that by cutting down the FeO, the requirement of SiO_2 for FeO to form slag would be so great that there would be none left for the As to form speiss; and furnaces have been run successfully on this basis. On the other hand, Bretherton[1] has shown that increasing the CaO-content is very effective in counteracting the formation of speiss. In smelting two ores, one with Pb 3.5, Fe 6.5, Mn 2.5, Zn 7.0, CaO 21.0, MgO 11.7, SiO_2 8.5, Al_2O_3 1.0, As 3.5, S trace; the other with Pb 11.5, Fe 20.0, Zn 4.0, CaO 3.0, Mn none, MgO none, SiO_2 30, As trace, S trace, and making a slag running high in CaO and MgO, he produced no speiss whatever. The high content of MgO, of course, raised the formation temperature of the slag and

[1] *Tr. A. I. M. E.*, 1915, LII, 730.

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with it the temperature of the furnace, which in its turn greatly favored the volatilization of As through the open charge.

On account of the high melting-point of speiss, it is necessary to have a high temperature at the tuyères in order to prevent the speiss from chilling, especially as it floats on the lead and is covered by matte. With considerable amounts of speiss, it may be necessary to aim for a sesqui-silicate slag of high formation temperature.[1]

With a speiss-crust forming in the hearth of the furnace, scrap iron is frequently added to the charge to liquefy the crust. This is useful if the speiss does not contain enough Fe; it cannot do any good, if the accretion has been caused by an insufficient temperature at the tuyère-level, or by having kept too low the level of the lead in the crucible.

150. Calculation of Charge, General.[2]—In calculating a charge for the blast furnace there have to be considered the weights of charge and fuel, the slag best suited for the ore, the amount of lead the charge is to contain, the richness of the lead bullion that is to be produced, and the quantities of speiss, matte, and slag that will ensue. A complete calculation will give full information on all these points.

The weight of charge to be introduced at a time into the furnace is governed in part by the amount of fuel required to smelt it, and the latter by the throat-area. In hand-feeding charges which contain fine ores, the smallest amount that may be used is that which completely covers the preceding charge so as to form a distinct layer. Hahn[3] ascertained the amount by placing on the feed-floor a wooden frame of the same area as the throat, filling it with coke to form a layer, and then weighing the coke. If charge and coke are mixed, there is danger of the heat creeping up through the porous coke;[4] if the two are kept separate,[5] the ascending gas entering the porous coke layer will be distributed evenly over the whole area, and act effectively upon the overlying layer of ore

[1] Henrich, *Eng. Min. J.*, 1883, XXXVI, 211.

[2] Murray, *Eng. Min. J.*, 1887, XLIV, 111, 1892, LIII, 281.
Newhouse, *School Min. Quart.*, 1887–88, IX, 373.
Furman, *op. cit.*, 1892–93, XIV, 134.
Furman, H. van F.-Pardoe, W. D., "Manual of Practical Assaying," Wiley, New York 1908, 367.
Saint Dizier, *Colo. State School Min. Sc. Quart.*, 1893, II[1], 50.
Hersam (Graphical Method), *Tr. A. I. M. E.*, 1901, XXXI, 340.
Chauvenet, *Min. Reporter*, 1907, LVI, 56, 76, 96, 148, 170, 190, 212, 264, 378, 396.
Chauvenet, R., "Chemical Arithmetic and Furnace Charges," Lippincott, Philadelphia, 1912, p. 207.
Richards, J. W., "Metallurgical Calculations," McGraw-Hill Book Co., New York, 1918, p. 593.
Earl, *Eng. Min. J.*, 1909, LXXXVII, 962.
von Schlippenbach, *Metall. u. Erz.*, 1915, XII, 399.
Dudley, *Met. Chem. Eng.*, 1917, XVI, 87, 129.

[3] *Tr. Inst. Min. Met.*, 1899–90, VIII, 267.

[4] Austin, *Min. Sc. Press*, 1907, XCIV, 61.

[5] Dwight, *Tr. A. I. M. E.*, 1902, XXXII, 363.

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and flux which, in the absence of coke, offers easy passages, but blankets the gases and compels them to force their way through.

With mechanical feeding, it is not practicable to feed coke and charge in layers, the two becoming more or less mixed. If the charge is coarse, as is the case when it contains ±70 per cent. of blast roasted material, an intimate mixing of coke and charge may even be advantageous,[1] as the charge is open throughout and permits an even ascent of gases.

The slag to be made will be governed by the character of the ore and the prices of fluxes that are available. The aim will be to acquire ores in such proportions that, when bedded or mixed, they will require the smallest amount of flux to form a slag of given composition. At present, very little barren iron ore is added to a charge. Some smelteries adhere to the typical slags given in Tables 64 and 65; others deviate more or less from them as seen in Table 53.

The amount of Pb in a charge, expressed in percentage of the sum of ore and flux, varies greatly. With furnaces having an Arents siphon-tap, it covers a range of from 6.5 to 40.0 per cent. With pure ore containing little or no Zn, As, or Sb, and not much S, it is safe to go as low as 8 per cent.; if these impurities are present to any extent, the charge should contain not less than 12 per cent. Thus, with silver-lead smelteries in which Pb mainly forms the means of collecting the Ag, the Pb in the charges ranges from 12 to 15 per cent. The lead plants of the Mississippi Valley (Federal, Collinsville, Herculaneum) and the silver-lead works of East Helena, Mont.; Northport, Wash.; Kellogg, Idaho; and Trail, B. C., run charges with from 30 to 40 per cent. Pb. Charges running high in Pb have to be run somewhat differently from those that contain little Pb. The loss by volatilization is greater with a charge low in Pb than with one that is high; the loss in Ag depends mainly on the loss of Pb.

151. Calculation of Charge by Algebraic Method.—The ore for which the charge is to be calculated is a carbonate containing some galena. Its composition is given in Table 68. A few assumptions have to be made in the calculation in order to simplify the work.

The S-content is supposed to be all sulphide-S. If part of this S were sulphate-S, a correction would have to be applied, as in the presence of both sulphide- and sulphate-S, or even of sulphate-S alone, from 15 to 40 and even 50 per cent. of the S passes off as SO_2 and does not enter the matte.

All the As is accounted for in the speiss, whereas ore blast furnaces in our days are usually run in such a manner as to avoid the formation of speiss, most of the As passing off with the gases, and some entering the lead bullion and matte.

TABLE 68.—ANALYSIS OF MIXED SULPHIDE AND CARBONATE SILVER-LEAD ORE

SiO_2	FeO	MnO	CaO	MgO	BaO	ZnO	Al_2O_3	S	As	Pb	Cu	Ag, oz.	Au, oz.
32.6	14.8	4.3	2.2	5.26	1.5	2.4	2.5	4.4	0.5	20.7	2.9	50.5	Trace

[1] Anderson, *Eng. Min. J.*, 1916, CI, 885.

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TABLE 69.—CORRECTED ANALYSES OF CHARGE-COMPONENTS

	SiO_2	FeO	CaO	ZnO	Al_2O_3	S	As	Pb	Cu	Ag, oz.	Au, oz.
Lead ore	32.6	19.1	10.16	2.4	2.5	4.4	0.5	20.7	2.9	50.5	Trace
Iron ore	4.3	74.1	3.10	.....	.....	.....	.....	.....	.....	.....	.....
Limestone	2.7	4.5	53.96	.....	.....	.....	.....	.....	.....	.....	.....
Coke-ash	40.3	26.5	10.26	.....	20.4	.....	.....	.....	.....	.....	.....

The Zn present is figured as entering the slag, in which it replaces an equivalent amount of CaO, though in reality perhaps only 80 per cent. does this; the rest being volatilized or entering the lead bullion, matte and flue-dust.

No account is taken of the losses in Pb, Ag and Au. The three metals are assumed to be collected in the lead bullion, though as a matter of fact, one part is lost, and another goes into intermediary products, especially matte and flue-dust.

It will be noted that the analysis shows no F. If this were present, a certain amount of Ca, equivalent to that required by F to form CaF_2, would be rendered unavailable as basic flux.

The actual deviations from assumed figures vary with the character of ore and slag, and the working of the furnace, so that the results of a computation usually do not correspond accurately with those obtained from the furnace, at least with a new ore. The slight deviations are, however, easily corrected when the weights and analyses of the first products have been obtained.

The slag chosen for the charge is the type E (La Pise-Eilers), Table 64, with SiO_2 30, FeO 40, CaO 20, RO 10 per cent.

The charge shall weigh 1000 lb. and contain 10 per cent. foul slag of the same composition as the one that is to be formed. The weight of the coke shall be 15 per cent. of that of the charge.

The two fluxes used are iron ore and dolomitic limestone. The analysis of the former shows: SiO_2 4.3, FeO 72.4, MnO 1.7, CaO 3.1 per cent.; that of the latter: SiO_2 2.7, FeO 4.5, CaO 37.3, MgO 11.9 per cent.

The coke contains 10 per cent. ash, which consists of SiO_2 40.3, FeO 26.5, CaO 6.9, MgO 2.4, Al_2O_3 20.4 per cent.

Before beginning the calculation, it is necessary to bring the different slag-forming components of ore, flux, and coke under the three main heads of SiO_2, FeO, and CaO.

Ferrous and manganous oxides, FeO and MnO, have nearly the same molecular weights, 72 and 71, hence the two oxides are simply added. In the decomposition of PbS, metallic Fe is the principal reagent, hence FeO has to be changed into Fe as shown by $FeO \times \frac{7}{9} = Fe$, or $FeO \times 0.777 = Fe$.

Magnesia, MgO, and baryta, BaO, are generally figured as replacing equivalent amounts of CaO, although some metallurgists object to this procedure: $CaO : MgO = 56 : 40$, $CaO = MgO \times 1.4$; in the same manner, $CaO = BaO \times 0.4$.

Zinc oxide, ZnO, is figured as replacing CaO in the slag: $CaO = ZnO \times 0.7$, thus cutting down the CaO-content of the slag with the increase of ZnO.

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The analyses of ore, flux, and coke-ash, changed as indicated, are given in Table 70.

In figuring the charge, four preliminary calculations have to be made:

1. The amount of available FeO and Fe in the iron ore.
2. The amount of Fe required by the As to form the supposed Fe_5As.
3. The amount of Fe required to combine to FeS with the S not taken up by the Cu as Cu_2S.
4. The amount of flux required for the 15 lb. of ash in 150 lb. coke.

Available FeO and Fe in Iron Ore.—In the slag, 30 SiO_2 require 40 FeO. In 100 lb. iron ore there are 4.3 lb. SiO_2. These require

$$SiO_2 : FeO = 30 : 40 = 4.3 : x,$$
$$x = 5.7 \text{ FeO}.$$

The iron ore contains 74.1 per cent. FeO; deducting 5.7, gives 68.4 lb. available FeO, or $\frac{7}{9} \times \text{FeO} = 53.2$ lb. available Fe, in 100 lb. iron ore.

Arsenic and Iron.—100 lb. lead ore contain 0.5 lb. As.

$$As : 5Fe = 75 : 280 = 0.5 : x,$$
$$x = 1.86 \text{ Fe}.$$

How much iron ore will furnish this?

$$\text{Iron ore} : \text{available Fe} = 100 : 53.2,$$
$$y = 3.5 \text{ lb. iron ore}.$$

Copper, Iron, and Sulphur.—100 lb. lead ore contain 2.9 lb. Cu.

$$Cu_2S : S = 126.8 : 32 = 2.9 : x,$$
$$x = 0.73 \text{ S}.$$

Of the 4.4 lb. S contained in 100 lb. lead ore, 0.73 lb. is required for the Cu; the difference, $4.4 - 0.73 = 3.67$ lb., must be combined with Fe.

$$S : Fe = 32 : 56 = 3.67 : y,$$
$$y = 6.42 \text{ Fe}.$$

How much iron ore will furnish this?

$$\text{Iron ore} : \text{available Fe} = 100 : 53.2 = z : 6.42,$$
$$z = 12 \text{ lb. iron ore}.$$

There are required for the As and S in 100 lb. lead ore, $3.5 + 12.1 = 15.6$ lb. iron ore. These contain 0.67 lb. SiO_2, 11.56 lb. FeO, and 0.48 lb. CaO. Only the non-available FeO, that is, the FeO in combination with SiO_2 in the ratio of 30 SiO_2 : 40 FeO, enters the slag, viz., 0.89 lb.; the remaining 10.66 lb. available FeO having been used up as Fe to combine with As and S to form speiss and matte.

The 0.67 lb. SiO_2 have to be supplied with CaO in the ratio of 30 SiO_2 : 20 CaO:

$$SiO_2 : CaO = 30 : 20 = 0.67 : x,$$
$$x = 0.45 \text{ lb. CaO}.$$

This amount may be considered to be balanced by the 0.48 lb. CaO present in the 15.6 lb. iron ore. If this were not the case, the 0.44 lb. CaO would have to be supplied from limestone.

$$\text{Limestone} : CaO = 100 : 54.0 = y : 0.45.$$

Furnace-men calculate the amounts of FeO and CaO necessary to flux the SiO_2 of a coke-ash or a silicious lead ore either by the algebraic method, using equations with two unknowns, or by the approximate system using proportions, which is the more common mode of procedure.

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The *Algebraic Method* is taken up first.

Coke-ash.—Starting with 100 lb. coke-ash, the amounts of iron ore (x) and limestone (y) necessary for fluxing SiO_2 in the ratio of 30 SiO_2 : 40 FeO : 20 CaO can be found by expressing the amounts of FeO first in terms of CaO, then in terms of SiO_2, and finally putting these quantities equal to each other, when x and y can be easily calculated.

Let $SiO_2 = c$, $FeO = a$, $CaO = b$; then FeO : CaO $= a : b$, $FeO = \frac{a}{b} \times CaO$; and FeO : $SiO_2 = a : c$, $Fe = \frac{a}{c} \times SiO_2$.

The necessary analytical data are given in Table 70.

TABLE 70.—CORRECTED ANALYSES OF COKE-ASH AND FLUXES

Desired amount	Material	SiO_2	FeO	CaO
100	Coke-ash	40.3	26.5	10.3
x	Iron ore	4.3	74.1	3.1
y	Limestone	2.7	4.5	54.0

$$(1)\quad FeO = \frac{a}{b} CaO,$$

$$26.5 + 0.741x + 0.045y = \frac{40}{20}(10.26 + 0.031x + 0.54y),$$

$$x = 1.524y - 8.81.$$

$$(2)\quad FeO = \frac{a}{c} SiO_2,$$

$$26.5 + 0.741x + 0.045y = \frac{40}{30}(40.3 + 0.043x + 0.027y),$$

$$x = 39.77 - 0.013y.$$

$$(3)\quad 1.524y - 8.81 = 39.77 - 0.013y.$$

$$y = 31.6 \text{ lb. limestone},$$

$$x = 39.4 \text{ lb. iron ore}.$$

Lead Ore.—Starting with 100 lb. lead ore, the necessary analyses are given in Table 71.

TABLE 71.—CORRECTED ANALYSES FOR LEAD ORE AND FLUXES

Desired amount	Material	SiO_2	FeO	CaO
100	Lead ore	32.6	19.1	10.2
x	Iron ore	4.3	74.1	3.1
y	Limestone	2.7	4.5	54.0

$$(1)\quad FeO = \frac{a}{b} CaO,$$

$$19.1 + 0.741x + 0.045y = \frac{40}{20}(10.16 + 0.031x + 0.539y),$$

$$x = 1.8 + 1.52y.$$

$$(2)\quad FeO = \frac{a}{c} SiO_2,$$

$$19.1 + 0.741x + 0.045y = \frac{40}{30}(32.6 + 0.043x + 0.027y),$$

$$x = 35.63 - 0.013y.$$

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(3) $1.8+1.52y=63.46-0.013y$.

$y=22.4$ lb. limestone,

$x=35.3$ lb. iron ore.

Summing-up.—In making up the desired charge of 1000 lb., there have to be taken into account, first, 15 lb. coke-ash requiring 5.9 lb. iron ore and 4.7 lb. limestone, and 100 lb. foul slag, which makes a total of 125.6 lb.; secondly, the difference $1000-125.6=874.4$ lb., which is to be made up of lead ore and its fluxes. Now 100 lb. lead ore were found to require 35.3 lb. iron ore and 22.4 lb. limestone for slagging the SiO_2, and 15.6 lb. iron ore for combining with As and S; this gives a total of 173.3 lb. The factor x, by which this weight has to be multiplied to give 874.4 lb., is obtained by $173.3x=875$; $x=5.046$. This gives the charge: coke-ash, 15 lb. (150 lb. coke); foul slag, 100 lb.; lead ore, 504.6 lb.; iron ore for SiO_2, 178.1 lb.; iron ore for As and S, 78.7 lb.; limestone, 113 lb.; which makes, with the iron ore and limestone required for the coke-ash a total of 1000 lb. This is shown in Table 72.

TABLE 72.—SUMMARY OF CALCULATED CHARGE

Material		SiO_2		FeO		CaO		ZnO		Al_2O_3	
Name	Dry weight, lb.	Per cent.	Lb.	Per cent.	Lb.	Per cent.	Lb.	Per cent.	Lb.	Per cent.	Lb.
Coke-ash	15.0	40.3	6.0	26.5	4.0	10.3	1.5	.....	....	20.4	3.1
Slag	100.0	30.0	30.0	40.0	40.0	20.0	20.0	.....	....	.....	.....
Lead ore	504.6	32.6	165.5	19.1	96.4	10.2	51.3	2.4	12.1	2.5	12.6
Iron ore (SiO_2)	184.0	4.3	7.9	74.1	136.3	3.1	5.7	.....	....	.....	.....
Iron ore (As, S)	78.7	4.3	3.4	74.1	58.3	3.1	2.4	.....	....	.....	.....
Limestone	117.7	2.7	3.2	4.5	5.3	54.0	63.5	.....	....	.....	.....
Totals	1,000.0	.....	216.0	.....	340.3	.....	144.4	.....	12.1	.....	15.7

Material		Ag		Pb		As		Cu		S	
Name	Dry weight, lb.	Oz. per ton	Oz.	Per cent.	Lb.	Per cent.	Lb.	Per cent.	Lb.	Per cent.	Lb.
Coke-ash	15.0	.....	.....	.....	.....	.....	.....	.....	....	.....	....
Slag	100.0	.....	.....	.....	.....	.....	.....	.....	....	.....	.....
Lead ore	504.6	50.5	12.8	20.7	104.5	0.5	2.5	2.9	14.6	4.4	22.2
Iron ore (SiO_2)	184.0	.....	.....	.....	.....	.....	.....	.....	....	.....	.....
Iron ore (As, S)	78.7	.....	.....	.....	.....	.....	.....	.....	....	.....	.....
Limestone	119.7	.....	.....	.....	.....	.....	.....	.....	....	.....	.....
Total	1,000.0	.....	12.7	.....	104.5	.....	2.5	.....	14.6	.....	22.2
Factor	0.1385	104.5 lb. lead bullion, assaying 243 oz. per ton				11.8 lb. speiss		69.9 lb. matte			

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If the slag ratio is to be SiO_2 30 : FeO 40 : CaO 20, the 216 lb. SiO_2 will call for $216 \times \frac{4}{3} = 288$ lb. FeO, and $216 \times \frac{2}{3} = 144$ lb. CaO in the slag. The CaO is seen to balance the corresponding column in the table, but there is left a balance of $340.3 - 288 = 52.3$ lb. FeO in excess. This will be required to furnish the Fe for the speiss and matte.

In 1000 lb. charge there are contained 104.5 lb. or 10.45 per cent. Pb. This 104.5 lb. Pb collects the 12.7 oz. Ag, hence the lead bullion will assay 243 oz. Ag per ton, presupposing that all the Ag has entered the Pb.

The charge contains 2.5 lb. As which with 9.3 lb. Fe forms 11.8 lb. speiss.

The 14.6 lb. Cu of the charge, requiring 3.7 lb. S, will form 18.3 lb. Cu_2S. Deducting the 3.7 lb. S from the total of 22.2 lb., leaves 18.5 lb. which, with 32.4 lb. Fe, give 50.9 lb. FeS. The total matte formed will be $18.3 + 50.9 = 69.2$ lb. This presupposes that the matte is free from Pb; actually it contains 10–12 per cent. Pb. Table 72 further shows that there are 10 per cent. slag and 15 per cent. coke on the charge.

It will be noticed that in summing up the results of the calculation, the weight of the coke-ash (15 lb.) has been included, although not that of the coke. This is not usually done by lead and copper smelters who, contrary to the practice of the iron smelter, include only ores and fluxes, and not the fuel. In this instance the coke-ash has been added for the purpose of bringing together in the table everything that influences the formation of the slag.

In making up the charge for the blast furnace, the moisture has still to be considered. If the lead ore contains, for instance, 5 per cent. of moisture, 531 lb. of moist ore will have to be used to correspond to 505 lb. of dry ore:

Moist Ore : Dry Ore $= 100 : 95 :: x : 510$; $x = 531$.

The same is the case with fluxes and fuel.

It is to be noted that figuring a charge according to the algebraic method has one great advantage over the method next to be described, viz., that it shows in what proportions any three classes of silicious, ferruginous, and calcareous ores are best mixed so as to become self-fluxing.

152. Calculation of Charge by the Method of Proportions.—Ore, flux, fuel, and slag are the same as in the preceding computation.

The preliminary calculations, such as bringing the different components of ore, flux, and fuel under the heads of SiO_2, FeO, and CaO, are made as before. The total weight (1,000 lb.) that the charge is to have, and with it the percentage of fuel (15 per cent.) and slag (10 per cent.) to be added are fixed. The available FeO and Fe of the iron ore are determined as in §151.

Two calculations are now necessary to determine the amounts of iron ore and limestone required by the coke-ash and by the ore.

Coke-ash.—The analyses of the ash and the two fluxes, iron ore and limestone, are entered in Table 73. The 150 lb. of coke contain 15 lb. of ash; for these the totals of SiO_2, FeO, and CaO are figured and entered in the table.

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There are present 6.04 lb. of SiO_2; how much FeO is required?

$$SiO_2 : FeO : : 30 : 40 : : 6.0 : x;$$
$$x = 8.0 \text{ lb. FeO necessary;}$$
$$4.0 \text{ lb. FeO is present.}$$

The difference, $y = 4.0$ lb. FeO to be added.

To find the necessary iron ore:

$$\text{Iron ore} : \text{Available FeO} : : 100 : 68.4 : : z : 4.0,$$
$$z = 5.9 \text{ lb. iron ore.}$$

They are entered in the table; their total pounds of SiO_2, FeO, and CaO are figured and also entered.

To the previous amount of 6.0 lb. SiO_2 has been added, by the iron ore, 0.3 lb. SiO_2, making the total SiO_2, 6.3 lb., for which limestone has to be provided. How much CaO is required?

$$SiO_2 : CaO : : 30 : 20 : : 6.\dot{3} : u,$$
$$u = 4.2 \text{ lb. CaO necessary;}$$
$$1.7 \text{ lb. CaO is present.}$$

The difference, $v = 2.5$ lb. CaO to be added.

To find the necessary limestone (neglecting the small amounts of SiO_2 and FeO it contains):

$$\text{Limestone} : CaO : : 100 : 54.0 : : w : 2.5,$$
$$w = 4.6 \text{ lb. limestone,}$$

which are entered with the pounds of CaO they bring to the slag.

TABLE 73.—CALCULATION OF FLUXES FOR COKE-ASH

Material		SiO_2		FeO		CaO	
Name	Dry weight, lb.	Per cent.	Lb.	Per cent.	Lb.	Per cent.	Lb.
Coke-ash	15.0	40.3	6.0	26.5	4.0	10.3	1.5
Iron ore	5.9	4.3	0.3	74.1	4.4	3.1	0.2
Limestone	4.6	2.7	.....	4.5	.....	54.0	2.5
Total	25.5	.....	6.3	.....	8.4	.....	4.2

The weights of iron ore (5.9 lb.) and limestone (4.6 lb.) are practically the same as those found by the algebraic method (5.9 and 4.7).

For the required slag ratio of SiO_2 30 : FeO 40 : CaO 20, there would then be required to accompany the SiO_2, $6.3 \times \frac{4}{3} = 8.4$ lb. FeO, and $6.3 \times \frac{2}{3} = 4.2$ lb. CaO, which check with the amounts shown in the table.

Deducting 125.5 lb. (the sum of the slag and coke-ash with its iron ore and limestone), from the total weight of the charge of 1000 lb. gives 874.5 lb. as compared with 874.4 lb., as calculated in §151, to be made up by the ore and its fluxes.

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Lead Ore.—Table 74 is laid out for the ore; the analytical data are entered, and the calculation is made on a basis of 100 lb.

1. The amounts of iron ore required (15.6 lb.) by the As and S are calculated as in §151, and the results are entered in the table.

2. The 100 lb. of ore contain 32.6 lb. SiO_2, for which the necessary FeO has to be provided:

$$SiO_2 : FeO :: 30 : 40 :: 32.6 : x$$
$$x = 43.5 \text{ lb. FeO necessary;}$$
$$19.1 \text{ lb. FeO is present.}$$

The difference, $y = 24.4$ lb. FeO to be added.

To find the necessary iron ore:

$$\text{Iron ore} : \text{Available FeO} :: 100 : 68.4 :: z : 24.34$$
$$z = 35.5 \text{ lb. iron ore.}$$

3. To the 32.6 lb. SiO_2 of the ore have been added from the two additions of iron ore, $0.7 + 1.5 = 2.2$ lb. SiO_2, making the total of 34.8 lb. of SiO_2, for which CaO has to be provided:

$$SiO_2 : CaO :: 30 : 20 :: 34.8 : u;$$
$$u = 23.2 \text{ lb. CaO necessary;}$$
$$11.8 \text{ lb. CaO is present.}$$

The difference, $v = 11.4$ lb. CaO to be added.

To find the necessary limestone (neglecting the small amounts of SiO_2 and FeO it contains):

$$\text{Limestone} : CaO :: 100 : 54.0 : w : 11.4;$$
$$w = 21.1 \text{ lb. limestone,}$$

which is entered upon the table.

Checking the calculation from the slag ratio, SiO_2 30 : FeO 20 : CaO 20, shows for 34.8 lb. SiO_2 a requirement of $34.8 \times \frac{4}{3} = 46.4$ lb. FeO, and of $34.8 \times \frac{2}{3} = 23.2$ lb. CaO. The CaO checks with the figure in the table; the FeO leaves an excess of $57.0 - 46.4 = 10.6$ lb. FeO, which is equivalent to 8.2 lb. Fe, and is needed to supply the 1.9 Fe for the speiss and the 6.4 lb. Fe for the matte.

TABLE 74.—CALCULATION OF FLUXES FOR LEAD ORE

Material		SiO_2		FeO		CaO	
Name	Dry weight, lb.	Per cent.	Lb.	Per cent.	Lb.	Per cent.	Lb.
Ore	100	32.6	32.6	19.1	19.1	10.2	10.2
Iron ore for As and S.	15.6	4.3	0.7	74.1	11.6	3.1	0.5
Iron ore for SiO_2	35.5	4.3	1.5	74.1	26.3	3.1	1.1
Limestone	21.1	2.7	.....	4.5	.....	54.0	11.4
Total	172.2	.....	34.8	.....	57.0	.....	23.2

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The figure found for iron ore, 35.5, checks sufficiently close with the 35.3 lb. found by the algebraic method; that for limestone is slightly lower (21.1 *vs.* 22.4 lb.), as the SiO_2 and FeO contained in the limestone have been neglected, making the available CaO 54.0, which is slightly too high.

If the items of the ore charge are now multiplied by 5.046 (as in §151) and those of the coke charge added, the sum of 1000 lb., the entire charge, will again be obtained.

153. Chemistry of the Blast Furnace, General.—This subject has not yet been fully studied; a theoretical discussion is contained in Guyard's paper "Argentiferous Lead Smelting at Leadville."[1]

The two leading processes that take place in the blast furnace are reduction and precipitation, and incidentally sulphurization, at temperatures ranging from 1100 to 1200° C. at the tuyère-level to 150° at the throat of the furnace. The charge, ore, flux and coke, fed at the throat, is subjected in its passage down the shaft during about 6 hr., at gradually increasing temperatures, to chemical reactions and physical changes which result in transforming it into lead bullion, speiss, matte, slag, and gases; the liquid products are collected, and the gases pass off into the open.

The principal reducing agents are C and CO.

The C is derived from the coke. It acts upon metallic oxides as soon as with rise of temperature its affinity for O is greater than that of the metal combined with O; it begins to act at about 400° C., and its affinity for O increases with the temperature. The product of oxidation is CO_2, if it acts upon an oxide easy of reduction, say below 1000° C.; it is CO, if the oxide is difficult of reduction, say above 1000° C.: $2PbO+C=Pb_2+CO_2$; $Fe_xO_{y+1}+C=Fe_xO_y+yCO$; $Fe_xO_y+yC=xFeO+yCO$.

The CO is derived from the combustion of C at an elevated temperature: $2C+O_2=2CO$; from the action of C upon Met. O above 1000° C., and upon CO_2 at any temperature above 400° C. The reducing power of CO, which begins at above 200° C., is favored by an increase of temperature to about 1000° C., and then falls off very quickly.

The leading precipitating agent is Fe; other metals, however, may act in a similar manner if they have greater affinities for S than the Met. S to be decomposed. Fournet[2] placed the leading metals according to their affinities for S in the following order:

Fournet Series: Cu—Fe—(Co—Ni)—Sn—Zn—Pb—Ag—Hg—Au—As—Sb. The more recent work of Schütz[3] gives:

Schütz Series: Mn—Cu—Ni—Fe—Sn—Zn—Ag.

Thus Schütz has reversed the order of Fe and Ni of the Fournet series, and has added Mn. In lead smelting, MnO_x is not reduced to the metallic state and can be left out of consideration as far as precipitation is concerned.

[1] Emmons, "Geology and Mining Industry of Leadville," monograph XII, *U. S. Geol. Survey*, Washington, 1886, p. 731.

[2] *Ann. Min.*, 1833, IV, 3, 225.

[3] *Metallurgie*, 1907, IV, 659, 694.

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Precipitation, or decomposition of Met. S by metal, begins at about 900° C.;[1] the velocity of reaction increases with the temperature and reaches its maximum at tuyère-level, or at from 1100 to 1200° C. The reversibility of the reaction $Met'S+Met''\rightleftarrows Met'+Met''S$ counteracts the complete decomposition of Met'S.

The behaviors of the leading components of a lead charge under furnace conditions have been outlined in §21–41 and §134–150. There remain to be considered the changes Ag_2S and AgCl (BrI) may undergo, and the effects mixtures of CO_2 and CO in the ascending gas current have upon C and Fe_xO_y, as well as of C upon Fe_xO_y.

Ag_2S and Ag, which melt at 815 and 960° C. respectively, form at 903° mixtures containing from 17.25 to 94.25 per cent. Ag, which separate into two layers,[2] and at 804° C. an eutectic with 99 per cent. Ag_2S. The sulphide, Ag_2S, heated with PbO is readily decomposed: $Ag_2S+2PbO=2(Pb.Ag)+SO_2$;[3] heated with S-ides it is likely to form eutectic mixtures and solid solutions (see Matte, §179); heated with metals having a stronger affinity for S than Ag, it is decomposed only in part, for similar reasons. Thus Mostowitsch[4] formulated its behavior with Pb by $2Ag_2S+2Pb=Ag_2Pb+Ag_2S.PbS$.

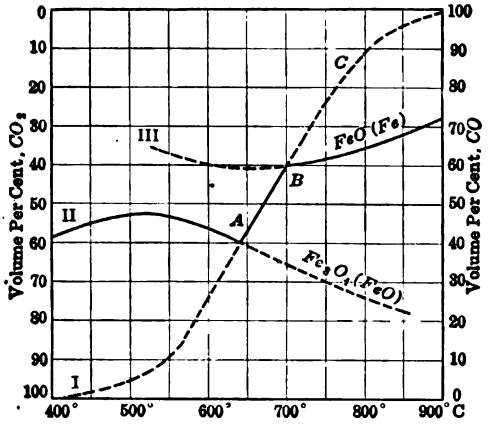

FIG. 324.—Equilibrium diagram of Fe_xO_y and C in mixtures of CO and CO_2 (Bauer-Glässner, Boudouard.)

The chloride, AgCl, which is readily volatile, is reduced to the metallic state when heated with Zn or Fe or Cu or Pb or Sn or Sb or Bi.[5]

Au follows Ag in most fire-processes.

The reaction $CO_2+C\rightleftarrows 2CO$ begins at 550° C., but is likely to be incomplete on account of the reversibility of the equation. The research of Boudouard[6] has shown that reduction stops at 650° C. with a gas-mixture of CO_2 61+CO 39 per cent. vol.; at 800° with CO_2 7+CO 93 per cent.; at 950° with CO_2 4+CO 96 per cent. His equilibrium diagram for different volumes of CO_2 and CO in contact with C is shown in curve I of Fig. 324. If the curve is read from right to left, it will give the conditions of equilibrium of the ascending gas current.

At 900 or say 1000° C., no CO_2 can be in equilibrium with CO in the presence

[1] Schütz, *loc. cit.*

[2] Bissett, *Tr. Chem. Soc.*, 1914, CV, 1223; *J. Inst. Met.*, 1914, XII, 293.

[3] Percy, J., "Metallurgy of Silver and Gold," Murray, London, 1880, pt. 1, p. 38.

[4] *Met. Chem. Eng.*, 1916, XIV, 703.

[5] Percy, *op. cit.*, p. 93.

[6] *Ann. chim. phys.*, 1910, XXIV, 28.

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of a sufficient amount of C, hence the C at the tuyères ought to burn to CO. If, nevertheless, much CO_2 is formed in the lead blast furnace, this is caused by the small amount of C present at the tuyère-level, which cannot reduce all the CO_2 formed, and by the excess of air blown in, which oxidizes a large part of the CO formed. The mixture of CO_2 and CO rises quickly into the cooler regions above..

At 800° C., about 4 ft. above the tuyères, 10 per cent. vol. CO_2 is held in equilibrium as regards C by 90 per cent. vol. CO; at 550°, about 15 ft. above the tuyères, 90 vol. CO_2+10 vol. CO have no effect upon C. Below 500 or 400°, when CO in the presence of C cannot be in equilibrium with CO_2, the reaction of carbon deposition, $2CO = CO_2 + C$, may be expected with oxides difficult of reduction.

Curve II exemplifies the limits of the reaction $Fe_3O_4 + CO \leftrightarrows 3FeO + CO_2$. At 550° C. a gas of the composition 56 vol. CO_2+44 vol. CO is in equilibrium with Fe_3O_4. A gas with 60 vol. CO_2+40 vol. CO containing an excess of CO_2 over that demanded by the equilibrium will show a tendency to be changed into $56CO_2 + 44CO$, that is, to give off O; hence it will act oxidizingly upon Fe_3O_4. A gas of the composition 50 vol. CO_2+50CO, for analogous reasons, will have a reducing effect.

With regard to C, both gases will have an oxidizing bent at 550°.

Curve III in the same manner gives the limits of the reaction $FeO + CO \rightleftarrows Fe + CO_2$. At 750°, a gas of the composition 39 vol. CO_2+61 vol. CO will be in equilibrium with FeO. At points *A* and *B* only, that is at 680 and 700° C., can there exist a perfect equilibrium between C, CO, CO_2, Fe_3O_4(FeO), and FeO(Fe); at other temperatures it is disturbed. If the iron ore used as flux were solely Fe_3O_4, the ratio $\frac{CO}{CO_2}$ would have to represent values lying above the full-drawn lines.

In applying the foregoing to the chemistry of smelting in the lead blast furnace, it is convenient to consider separately the quickly ascending gas-current and the slowly descending ore-charge.

154. The Ascending Gas-current.—There have not been published any analyses of gases formed at the tuyères. Schertel[1] calculates from his analyses of gases passing off at the throats of Freiberg lead blast furnaces that at the tuyères C burns mainly to CO_2. Fig. 325 represents averages of a large number of gas analyses made in 1905 from a Colorado blast furnace, the slag of which gave with a Wanner pyrometer 1120° C. when flowing from the tap-hole. The abscissæ represent percentages volume of CO_2, CO, and O; the ordinates give in feet the vertical distances from the tuyères at which the sampling-tube was inserted. At 1 ft. above the tuyères the curves show 8 vol. CO, 12CO_2 and 0.2 free O. Table 75 gives analyses of waste gases passing off at the throats of furnaces at Freiberg, Saxony, and at Denver and Pueblo, Colo. At the Freiberg works the ore-charge consisted of slag-roasted lead ore, slag, and burnt pyrite; the slags formed contained 4.75 per cent. CaO and 0.54 per cent.

[1] *Freib. Jahrb.*, 1880, 37; *Berg. Hüttenm. Z.*, 1880, xxxix, 85.

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MgO, so that the CO_2 from dolomitic limestone present could not have much influence upon the CO_2-content of the gas. At the Colorado works, bedded mixtures of raw and roasted ores were smelted with high-CaO slags.

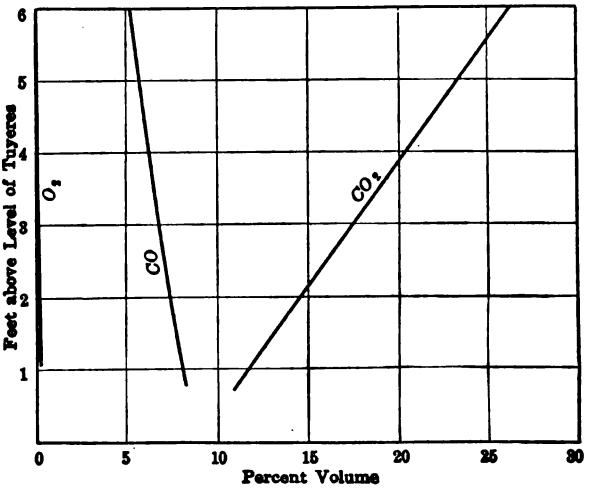

FIG. 325.—Percentages of O_2, CO_1 and CO_2 in lead blast furnace gas at different heights above tuyère-level.

TABLE 75.—ANALYSES OF WASTE GASES OF LEAD BLAST FURNACES

	Freiberg, Saxony (*a*)						Denver, Colo. (*b*)				Pueblo, Colo. (*c*)			
	Ore smelting			Slag smelting			Globe plant				Eilers plant			
N.....	72.73	75.30	75.20	75.5	76.0	74.8	74.1	69.4	73.0	68.6	77.6	78.9	79.6	75.5
CO_2.....	16.26	17.80	17.20	16.6	17.4	18.5	15.2	17.6	17.4	21.6	18.8	19.6	16.6	18.2
CO.....	10.06	5.20	5.40	5.9	4.3	3.5	9.7	8.0	5.4	10.8	3.0	1.5	3.6	5.9
CH_4.....	0.36	0.10	0.70	2.0	1.8	0.3	.....	.....	.....	.....	....	....	....	....
H.....	0.59	1.60	1.50	.....	0.5	2.6	.....	.....	.....	.....	....	....	....	....
CO_2.....	61	77	76	74	80	84	61	69	76	66	86	93	82	76
CO.....	39	22	24	26	30	16	39	31	24	33	14	7	17	24

(*a*) Schertel, *loc. cit.* (*b*) Iles, M. W., private communication. (*c*) Dwight, A. S., private communication.

It will be noted that the excess of CO_2 over CO in the gases issuing from the throats of the furnaces, represented in Fig. 325, is much larger than that in the gases taken 1 ft. above the tuyères; hence C will have been largely burnt to CO_2 by contact with oxides easy of reduction, and CO will have acted more or less as a reducing agent. Table 75 shows that the ratio of CO_2 : CO with the low-CaO. slags of Freiberg is higher than with the high-CaO slags of Denver.

It may be assumed that at the temperature of the tuyère-region of 1100°–1200° C., the C of the coke will burn more to CO_2 than to CO. The mixture of CO_2, CO, and N (omitting possible H and SO_2) ascending in the furnace, shows, Fig. 325, an increase in CO_2 and a corresponding decrease in CO. At

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about 900°, the CO_2-content ought to be considerably increased by the dissociation of the limestone flux; but, if the straight-line curves represent true averages, the CO_2 acting upon C must have been in part reduced by the coke, as indicated by curve I in Fig. 324. On the other hand, solid C acting upon oxide difficult of reduction must have increased the CO-content. Just what are the mutual relations between the several reactions requires systematic investigation to determine.

Analyses of throat-gases from Altenau and St. Andreasberg, Harz Mountains, by K. Waldeck[1] show a great preponderance of CO over CO_2 as shown in Table 76.

TABLE 76.—ANALYSES OF GASES FROM LEAD BLAST FURNACES OF ALTENAU AND ST. ANDREASBERG.

Altenau			St. Andreasberg		
Sample, feet above tuyères	CO_2	CO	Sample, feet above tuyères	CO_2	CO
1.15	5.1	20.4	0.49	6.8	17.7
3.93	6.1	30.3	2.36	11.5	15.8
7.22	7.8	21.8	4.61	10.9	14.9
.....	1.5	1.1	6.56	8.1	12.2

This may be as ascribed to the practice of smelting raw galena concentrate with burnt pyrite as flux. Such a process requires a plentiful reduction of Fe_2O_3 to Fe accompanied by the formation of CO in preference to CO_2. With the large excess of CO over CO_2, there is less likelihood of the gas-mixture having any oxidizing effect upon the coke; on the other hand, it ought to furnish a greater reduction of metallic oxide than under reverse conditions.

A comparison of the gases from slag- and ore-smelting at Freiberg in Table 75 indicates that in the latter operation the precipitation process is more prevalent than in the former, in which ore-slag with from 3 to 6 per cent. Pb is put through the furnace with an addition of 20 per cent. imperfectly roasted matte furnished by the ore-furnace.

155. Descending Ore-charge.—In considering the changes in the descending ore-charge, it is convenient to distinguish four zones: The zone of preparatory heating, 100°–400° C.; the upper zone of reduction, 400°–700°; the lower zone of reduction, 700°–900°; and the zone of fusion, 900°–1,200°. There is no sharp line of demarcation between the zones, one passing over into the other; the reactions beginning in one zone may be finished only in the next or perhaps even the second lower down. The process of heat-interception is continuous from the tuyères to the throat, or from 1200°–100° C.

[1] "Gasanalytische Untersuchungen an Bleischachtöfen," Herman, Berlin, 1901, and "Streifzüge durch die Blei u. Silberhütten des Oberharzes," Knapp, Halle, 1907; abstract, Hofman, *Min. Ind.*, 1902, XI, 445.

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1. *Zone of Preparatory Heating*, 100°–400° *C.*—The charge fed into the furnace gives up its hygroscopic water, then that which is chemically combined; clayey material, limonites, and other hydrates give up at 400° only part of this water. The evaporation of water absorbs heat and thus assists in keeping the top of the furnace cool.

2. *Upper Zone of Reduction*, 400°–700° *C.*—The decomposition of hydrates continues, the dissociation of carbonates and of some sulphates begins. The reducing effect of CO, which theoretically begins at 200° C., becomes noticeable only at about 400°; and increases with rise of temperature up to about 900°. It will therefore be an active agent in reducing PbO completely to Pb, and $PbSO_4$ partly to PbS; it will also start the reduction of Fe_2O_3. The C of the coke, which begins to act reducingly at about 400° C. and increases as the temperature rises, will change PbO completely into Pb, and $PbSO_4$ partly into PbS; it will assist CO in its reduction of Fe_2O_3, and will transform CO_2 of the gas current into CO. There will further take place reactions of $PbSO_4$ and PbO with PbS, setting free Pb and SO_2.

3. *Lower Zone of Reduction*, 700°–900° *C.*—The reactions started in the preceding zone continue and are in part completed; the effect of C becomes more marked than that of CO. The $CaSO_4$ in blast roasted ore as well as $BaSO_4$ in raw ore are more or less reduced to CaS and BaS to be dissolved later on mainly by the slag. The dissociation of carbonates is about completed, $CaCO_3$ at 910°. The union of SiO_2 with unreduced PbO and $PbSO_4$ begins, as does the decomposition of PbS, $PbAs_x$, $PbSb_y$ by Fe; sulphurization of Cu begins about at the same temperature. Matte of eutectic composition begins to soften. Everything is prepared to be liquefied and to bring to completion the chemical processes.

4. *Zone of Fusion*, 900°–1200° *C.*—The reductions of oxides, including ZnO, and decompositions of S-ides are completed; ascending Zn-vapor is oxidized and sulphurized. Lead reduced in the upper parts of the furnace trickles through the charge, picking up Ag on its way, and acting possibly upon lead arsenate and antimonate; it joins the Pb set free lower down from PbS, $PbAs_x$, $PbSb_y$, and continues to take up Ag. The sulphides of eutectic composition, which softened higher up, become liquid and dissolve other sulphides to form matte with a melting-point lying below that of slag-formation. The slag-components SiO_2, FeO, and CaO form a slag of lowest formation temperature and this, trickling downward, dissolves the remaining SiO_2, FeO, CaO, as well as other bases such as Al_2O_3, ZnO, etc. Scorified PbO is set free and reduced by C to Pb. The three main products, lead, matte, and slag, settle in layers according to their specific gravities; lead passing downward through slag and matte robs these of some precious metal; matte in a similar way removes Pb, Cu, and Ag from the slag; at the contact planes of the products interchange of components takes place to a moderate degree with a tendency toward collecting sulphide in matte, and metal in lead. The lead passes off through the lead-well; speiss, matte, and slag are tapped into the fore-hearth, from which slag

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overflows into a waste-slag pot or granulating apparatus, and speiss and matte are tapped periodically into suitable receivers.

156. Thermal Balance, General.—The heat balance of a metallurgical process has its debit and credit columns as has the balance sheet of a commercial account. In the debit column are entered the heat-producing factors: sensible heat entering the furnace, heat generated by the combustion of fuel, and exothermic reactions of the charge; in the credit column, the heat-absorbing factors: heat carried off in liquid products and in gases and fumes, heat absorbed by endothermic reactions of the charge, and heat lost by radiation and conduction. A study of the data gives a thermal insight into the process.

The thermal balance to be cast is based on blast furnace data kindly furnished by Mr. William Allen Smith, Manager of the smeltery of the St. Joseph Lead Co., Herculaneum, Mo. The construction and metallurgical details of the furnace are given in Table 53. The basis of the calculation is that of 5,000 kg. of charge smelted with 590 kg. of coke.

157. Rational Analyses.—There were obtained ultimate analyses of raw materials and products. As no rational analyses were available, these had to be made up by calculation assisted by a general knowledge of the mineralogical character of the components of the charge. While the rational analyses given in Table 77 are not absolutely accurate, it is believed that they are approximately correct.

158. Distribution of Materials. *A. Calculation of Weights of Constituents of the Matte.*—The matte-fall is 12.3 per cent. of the weight of the charge. The weight of matte is therefore 12.3 per cent. of 5000 kg. or 615 kg. It contains (analysis, Table 53) 81.4 kg. Pb, 24.8 kg. Cu, 297.1 kg. Fe, 36.8 kg. Zn, and 139.4 kg. S.

Combining the four metals with the S necessary to give sulphides, gives 81.4 Pb+12.6S=94.0 kg. PbS; 24.8Cu+6.2S=31.0 kg. Cu_2S; 36.8Zn+18.1S=54.9 kg. ZnS. This leaves 139.4−(13.0+6.2+18.1)=102.1 kg. S to be combined with Fe to form FeS; or 102.1S+178.7Fe=280.8 kg. FeS. There are present in the matte 297.1 kg. Fe; deducting 178.7 gives 118.4 kg. Fe, which with 45.1 kg. O furnishes 163.5 kg. Fe_3O_4.

B. Assignment of Materials.—1. Sintered ore, 3,834 kg. This contains:

(*a*) PbS, 571.3 kg., of which 94.0 kg. (see above) goes into the matte, leaving 571.3−94.0=477.3 kg. PbS. Reducing this to Pb and S gives 413.0 kg. Pb going into metal and 64.3 kg. S, of which 4.3 kg. go to furnish the small percentage of S in the metal and 60.0 kg. enters the matte as FeS.

(*b*) $PbSO_4$, 214.7 kg., which resolved gives 146.7 kg. Pb going into the metal and 68.0S+O entering the gas.

(*c*) $2PbO.SiO_2$, 1054.4 kg. The 1920 kg. waste slag reported[1] carry 1.3 per cent. or 24.7 kg. Pb, which corresponds to 30.2 kg. 2 $PbO.SiO_2$. Deducting this amount from the total leaves 1054.4−30.2=1024.2 kg.

[1] The theoretical balance sheet of materials shows 2,241.8 kg., including 463.7 kg. of remelted slag, or 1,778.1 kg. as compared with the reported weight of 1,920 kg.

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TABLE 77.—THEORETICAL BALANCE SHEET OF MATERIALS OF ONE CHARGE OF 5000 KG. ORE+590 KG. COKE

Components	Charge: Chemical analysis per cent.	Charge: Weight, kg.	Products: To metal, kg.	Products: To matte, kg.	Products: To slag, kg.	Products: To gas, kg.
Sintered ore 3,834 kg.						
PbS	14.9	571.3	417.3	154.0	.....	.....
$PbSO_4$	5.6	214.7	146.7	.....	.....	68.0
$2PbO.SiO_2$	27.5	1,054.4	838.0	.....	151.6	64.8
PbO	3.9	149.5	138.8	.....	.....	10.7
ZnS	7.9	302.9	.....	94.7	111.4	96.8
$2FeO.SiO_2$	33.8	1,295.9	.....	220.6	1,025.4	49.9
CaO	3.2	122.6	.....	.....	122.6	.....
MgO	1.9	72.9	.....	.....	72.9	.....
Cu_2S	1.0	38.3	6.5	31.0	.....	0.8
X	0.3	11.5	.....	.....	11.5	.....
Burnt fume 17 kg.						
$PbSO_4$	88.5	15.1	10.3	.....	.....	4.8
PbO	3.2	0.5	0.5	.....	.....	.....
Fe_2O_3, SiO_2, X	8.3	1.4	.....	.....	1.4	.....
Lime rock 322 kg.						
$CaCO_3$	88.0	283.4	.....	.....	159.0	124.4
$MgCO_3$	4.4	14.1	.....	.....	6.7	7.4
$2FeO.SiO_2$	0.9	2.9	.....	.....	2.9	.....
SiO_2	4.2	13.5	.....	.....	13.5	.....
X	2.5	8.1	.....	.....	8.1	.....
Pyrites cinder 140 kg.						
FeS	8.5	11.9	.....	11.9	.....	.....
Fe_2O_3	62.7	87.8	.....	84.7	.....	2.9
SiO_2	6.9	9.6	.....	.....	9.6	.....
X	21.9	30.7	.....	.....	30.7	.....
Iron scale 32 kg.						
Fe_2O_3	94.4	30.2	.....	29.2	.....	1.0
SiO_2	4.6	1.5	.....	.....	1.5	.....
X	1.0	0.3	.....	.....	0.3	.....
Refinery skimmings 188 kg.						
PbO	100.0	188.0	174.6	.....	.....	13.4
Slag 467 kg.						
Slag	98.0	457.7	.....	.....	457.7	.....
Pb	2.0	9.3	3.3	.....	6.0	.....
Coke 590 kg.						
Fixed C	83.8	494.4	.....	.....	.....	494.4
Volatile H-C	1.6	9.4	.....	.....	.....	9.4
H_2O	6.3	37.2	.....	.....	.....	37.2
Ash	8.3	49.0	.....	.....	49.0	.....
Blast 5,868.5 kg. (5,819.5 dry).						
N	76.9	4,476.5	.....	.....	.....	4,476.5
O	23.1	1,343.0	.....	.....	.....	1,343.0
H_2O	.....	49.0	.....	.....	.....	49.0
Totals	.....	11,458.5	1,736.0	626.1	2,241.8	6,854.4

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silicate, which, reduced to component parts, gives 838.0 kg. Pb going into the metal, 121.4 kg. SiO_2 entering the slag, and 64.8 kg. O entering the gas.

(*d*) PbO, 149.5 kg.; reduced gives 138.8 kg. Pb going into the metal and 10.7 kg. O entering the gas.

(*e*) ZnS, 302.9 kg.; of this total there are found in the matte 54.9 kg., leaving 248.0 kg. which reduced give 166.0 kg. Zn+82.0 kg. S. The 1,920 kg. waste slag reported contain 2.2 per cent. S and 3.6 per cent. Zn or 42.2 kg. S and 69.2 kg. Zn. This leaves 39.8 kg. S to go to the matte as FeS, and 96.8 kg. Zn which will be assumed to be oxidized and carried out of the furnace in the gases.

(*f*) $2FeO.SiO_2$, 1295.9 kg., 60.0 kg. S go to the matte from the PbS, and this requires to make FeS, 105.0 kg. Fe, which is equivalent to 135.0 kg. FeO. 39.8 kg. S go to the matte from the ZnS, and this requires, to make FeS, 69.7 kg. Fe, which is equivalent to 89.5 kg. FeO. The Fe_3O_4 in the matte comes from the pyritic cinder, the iron scale, and the ferrous silicate. The 87.8 kg. Fe_2O_3 in the cinder correspond to 84.7 kg. Fe_3O_4+2.9 kg. O, and the 30.2 kg. Fe_2O_3 in the scale to 29.2 kg. Fe_3O_4+1.0 kg. O, giving a total of 84.9+29.2=114.1 kg. Fe_3O_4. In the matte there are present 163.5 kg. Fe_3O_4; the difference, 163.5 −114.1=49.4 kg. Fe_3O_4, has to come from the $2FeO.SiO_2$ in the sintered ore. Of the total FeO, 46.0 kg. FeO=49.4 kg. Fe_3O_4 go into the matte; the remainder, 1295.9−(135.0+89.5+46.0)=1025.4 kg. go into the slag. The 135.0+89.5=224.5 kg. of FeO, or 174.6 kg. Fe, go into the matte as 274.5 kg. FeS, securing the S from the PbS and the ZnS, leaving 49.9 kg. O to go to the gas.

(*g*) CaO, 122.6 kg.; MgO, 72.9 kg.; and X 11.5 kg. (undetermined) go into the slag.

2. Burnt fume, 17 kg.

(*a*) $PbSO_4$, 15.1 kg.; reduced gives 10.3 kg. Pb going into the metal, and 4.8 kg. S+O entering the gas.

(*b*) PbO, 0.5 kg.; neglecting the O, this gives 0.5 kg. Pb going into the metal.

(*c*) $Fe_2O_3+SiO_2$+X, 1.4 kg.; this enters the slag.

3. Lime rock, 322 kg.

(*a*) $CaCO_3$, 283.4 kg.; this gives 159.0 kg. CaO entering the slag, and 124.4 kg. CO_2 entering the gas.

(*b*) $MgCO_3$, 14.1 kg.; this gives 6.7 kg. MgO entering the slag, and 7.4 kg. CO_2 entering the gas.

(*c*) $2FeO.SiO_2$, 2.9 kg.; enters the slag.

(*d*) SiO_2, 13.5 kg.; enters the slag.

(*e*) X, 8.1 kg.; enters the slag.

4. Pyrite cinder, 140 kg.

(*a*) FeS, 11.9 kg.; enters the matte.

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(*b*) Fe_2O_3, 87.8 kg.; reduced gives 61.5 kg. Fe entering the matte, and 26.3 kg. O entering the gas.
(*c*) SiO_2, 9.6 kg.; enters the slag.
(*d*) X, 30.7 kg.; enters the slag.

5. Iron scale, 32 kg.
(*a*) Fe_2O_3, 30.2 kg.; reduced gives 21.1 kg. Fe entering the matte, and 9.1 kg. O entering the gas.
(*b*) SiO_2, 1.5 kg.; enters the slag.
(*c*) X, 0.3 kg.; enters the slag.

6. Refinery skimming, 188 kg.
PbO, 188 kg.; reduced gives 174.6 kg. Pb entering the metal, and 13.4 kg. O entering the gas.

7. Slag, 467 kg.
(*a*) Slag, 463.7 kg.; enters the slag.
(*b*) Pb, 3.3 kg.; enters the metal.

8. Coke, 590 kg.
(*a*) Fixed carbon, 494.4 kg.; enters the gas.
(*b*) Volatile hydrocarbon, 9.4 kg.; enters the gas.
(*c*) Water, 37.2 kg.; enters the gas.
(*d*) Ash, 49.0 kg.; enters the slag.

9. Blast, 5868.5 kg. (=5819.5 dry).

The furnace receives per minute 136 m^3 of air. It smelts 200 metric tons or 200,000 kg. charge in 24 hr., or 5000 kg., the basis of the thermal balance, in 36 min. The volume of air per charge of 5000 kg. is $36\times136=4896$ m^3. The atmosphere contains about 10 gm. moisture per m^3; the 4896 m^3 blower-air contain 49.0 kg. water. The air in the blower-room is assumed to be at 20° C.; the 4896 m^3 moist air at 20° C. and normal pressure correspond to 4500.8 m^3 dry air at 0° C.; these 4500.8 m^3 air weigh 5819.5 kg.=4476.5 kg. N+1343.0 kg. O. Adding the water gives 5819.5+49.0=5868.5 kg.

159. Calculation of Thermal Balance for a Charge.—In casting the thermal balance for one charge of 5,000 kg. ore and flux and 590 kg. coke, given in Table 77, there has to be ascertained the incoming heat, listed in Table 78 under Debit, and the outgoing heat, listed under Credit; the columns headed Per cent. of total show clearly the relations of heat-producing and heat-consuming factors.

The details of the calculations are as follows:

A. Debit Side of Balance Sheet.—1. Burning C to CO, 2430 Cal. per kg. C: The gas analysis gives $CO_2=19$ and $CO=10$ per cent. The total carbon in the gases is 494.4 kg. from the coke and 35.9 kg. from the lime rock. Of this, $\frac{10}{29}\times530.3=182.9$ kg. burns to CO. This leaves 347.4 kg. C in the gas as CO_2, of which 35.9 came from the lime rock, and 311.5 burned to CO_2 from the coke.

$$182.9\times2430=444{,}447 \text{ Cal.}$$

2. Burning C to CO_2, 8100 Cal. per kg. C:

$$311.5\times8100=2{,}523{,}150 \text{ Cal.}$$

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3. Burning Zn to ZnO, 1305 Cal. per kg. Zn: The zinc going in to the gas is 96.8 kg.

$$96.8 \times 1305 = 126{,}324 \text{ Cal.}$$

4. Burning FeO to Fe_3O_4, 341 Cal. per kg. FeO: Some FeO in the $2FeO.SiO_2$ of the sintered ore was assumed (page 347) to enter the matte as Fe_3O_4.

$$46.0 \times 341 = 15{,}686 \text{ Cal.}$$

5. Formation of FeS from Fe, 428 Cal. per kg. Fe:

$$174.7 \times 428 = 74{,}772 \text{ Cal.}$$

6. Formation of Slag, 135 Cal. per kg., approximately: The total slag produced is 2241.8 kg. There were obtained 1025.4 kg. from the $2FeO.SiO_2$, and 457.7 from foul slag, or 1483.1 kg. slagged material. Deducting this from the total gives $2241.8 - 1483.1 = 758.7$ kg. slag formed.

$$758.7 \times 135 = 102{,}425 \text{ Cal.}$$

7. Sensible Heat in Blast at 20° C.: This is found by the product of volume × specific heat × temperature.

Dry air $= 4500.8 \times 0.3035 \times 20 = 27{,}320$ Cal.

There are present 49.0 kg. $H_2O = 60.5$ m³ vapor.

$60.5 \times 0.343 \times 20 = 415$ Cal., giving $27{,}320 + 415 = 27{,}735$ Cal.

8. Sensible Heat in Charge and Coke at 20° C.: Figured as under 7 gives

$$5{,}000 \times 0.25 \times 20 = 25{,}000 \text{ Cal.}$$
$$\text{and } 590 \times 0.574 \times 20 = 6773 \text{ Cal.}$$

This finishes the heat-producing items to be entered in the Debit column. They show that 88.8 per cent. of the heat generated results from the oxidation of C.

B. Credit Side of Balance Sheet.—1. Reduction of PbS to Pb, 98 Cal. per kg. Pb: 414.0 kg. Pb in sintered ore + 3.3 kg. in slag = 417.3 kg.

$$417.3 \times 98 = 40{,}896 \text{ Cal.}$$

2. Reduction of $PbSO_4$ to Pb, SO_3, and O, 598 Cal. per kg. Pb: 146.7 kg. Pb in sintered ore + 10.3 kg. in burnt fume = 157.0 kg.

$$157.0 \times 598 = 93{,}886 \text{ Cal.}$$

3. Reduction of $2PbO.SiO_2$ to Pb, 425 Cal. per kg. Pb (estimate, no figures available):

$$838.0 \times 425 = 356{,}150 \text{ Cal.}$$

4. Reduction of PbO to Pb, 245 Cal. per kg. Pb: 138.8 kg. Pb in sintered ore + 0.5 kg. in burnt fume + 174.6 kg. in refinery skimming = 313.0 kg. Pb.

$$313.0 \times 245 = 76{,}905 \text{ Cal.}$$

5. Reduction of ZnS to Zn, 662 Cal. per kg. Zn: The Zn entering the gas as ZnO is first reduced from ZnS.

$$96.8 \times 662 = 64{,}082 \text{ Cal.}$$

6. Reduction of $2FeO.SiO_2$ to FeO, 154 Cal. per kg. FeO:

$46.0 \times 154 = 7084$ Cal. for the FeO that goes as Fe_3O_4 in to the matte.

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7. Reduction of $2FeO.SiO_2$ to Fe, 1371 Cal. per kg. Fe:

The Fe going in to the matte with the S of the PbS and ZnS amounts to 105.0+69.7=174.7 kg.

$$174.7 \times 1371 = 239{,}514 \text{ Cal.}$$

8. Reduction of Fe_2O_3 to Fe_3O_4, 97 Cal. per kg. Fe_3O_4:

84.7 kg. Fe_3O_4 in pyrite cinder+29.2 kg. in iron scale=113.9 kg. Fe_3O_4.

$$113.9 \times 97 = 11{,}048 \text{ Cal.}$$

9. Decomposition of $CaCO_3$, 806 Cal. per kg. CaO:

$$159.0 \times 806 = 128{,}154 \text{ Cal.}$$

10. Decomposition of $MgCO_3$, 733 Cal. per kg. MgO:

$$6.7 \times 733 = 4911 \text{ Cal.}$$

11. Reduction of Cu_2S to Cu, 160 Cal. per kg. Cu:

The metal carries 0.46 per cent. Cu or 6.5 kg. Cu.

$$6.5 \times 160 = 1040 \text{ Cal.}$$

12. Heat in metal at 800° C.; total heat of 25.1 Cal. per kg., approximately:

$$1736.0 \times 25.1 = 43{,}574 \text{ Cal.}$$

13. Heat in matte at 1200° C.; total heat of 250 Cal. per kg., approximately:

$$626.1 \times 250 = 156{,}525 \text{ Cal.}$$

14. Heat in slag at 1200° C.; total heat of 325 Cal. per kg., approximately:

$$2241.8 \times 325 = 728{,}583 \text{ Cal.}$$

15. Heat in gas at 225° C.:

182.9 C burn to CO, giving 338.7 m^3 (standard condition) with a mean specific heat of 0.309.

311.5 C burn to CO_2, giving 576.8 m^3 (standard condition) with a mean specific heat of 0.42.

4500.8 m^3 air contain 3565.0 m^3 N with a mean specific heat of 0.309.

49.0 kg. H_2O furnish 60.5 m^3 water vapor with a mean specific heat of 0.374.

CO	$338.7 \times 0.309 \times 225 =$	23,548 Cal.
CO_2	$576.8 \times 0.42 \times 225 =$	54,438 Cal.
N	$3{,}565 \times 0.309 \times 225 =$	247,880 Cal.
H_2O vapor	$60.5 \times 0.374 \times 225 =$	5,105 Cal.
Total heat in gas		=330,971 Cal.

16. Heat in cooling water:

Water per minute=560 kg.; water per charge=$560 \times 36 = 20{,}160$ kg.

Temperature of feed water 31.5° C., of overflow water 53.5° C.; rise in temperature 22° C.

$$20{,}160 \times 22 \times 1 = 443{,}520 \text{ Cal.}$$

17. Heat lost by radiation and conduction ascertained by difference.

This finishes the heat-absorbing items to be entered in the Credit column. They show what large amounts of heat are carried away by the metal products,

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the cooling water and the gases. The loss of heat by radiation and conduction of 18.5 per cent. appears somewhat high; it may be accounted for by some inacurate assumptions made in the calculation.

TABLE 78.—THERMAL BALANCE FOR ONE ORE-CHARGE OF 5000 KG.

Debit			Credit		
Items	Kilogram calories	Per cent. of total	Items	Kilogram calories	Per cent. of total
1. Burning C to CO	444,447	13.3	1. Reduction of PbS to Pb	40,896	1.2
2. Burning C to CO_2	2,523,150	75.5	2. Reduction of $PbSO_4$ to Pb	93,886	2.8
3. Burning Zn to ZnO	126,324	3.8	3. Reduction of $2PbO.SiO_2$ to Pb	356,150	10.6
4. Oxidizing FeO to Fe_3O_4	15,686	0.5	4. Reduction of PbO to Pb	76,905	2.3
5. Formation of FeS from Fe	74,772	2.2	5. Reduction of ZnS to Zn	64,082	1.9
6. Formation of slag	102,425	3.0	6. Reduction of $2FeO.SiO_2$ to FeO	7,084	0.2
7. Sensible heat in blast	27,735	0.8	7. Reduction of $2FeO.SiO_2$ to Fe	239,514	7.2
8. Sensible heat in charge and coke	31,773	0.9	8. Reduction of Fe_2O_3 to Fe_3O_4	11,048	0.3
			9. Decomposition of $CaCO_3$	128,154	3.8
			10. Decomposition of $MgCO_3$	4,911	0.1
			11. Reduction of Cu_2S to Cu	1,040	0.1
			12. Heat in metal	43,574	1.3
			13. Heat in matte	156,525	4.7
			14. Heat in slag	728,583	21.8
			15. Heat in gases	330,971	9.9
			16. Heat in cooling water	443,520	13.3
			17. Loss by radiation and conduction (by difference)	618,199	18.5
Total	3,346,312	100.0	Total	3,346,312	100.0

160. Smelting Operations, General.—The work around a blast furnace is conveniently taken up under the heads of blowing-in, work on the charging-floor, work on the furnace-floor, work on the dump, irregularities in the furnace, blowing-out, and furnace records.

161. Blowing-in, General.[1]—The operation comprises the three steps of warming the crucible, filling the furnace, and starting the smelting.

The warming of the crucible in a new furnace must be done slowly and with

[1] Harrison, *Eng. Min. J.*, 1911, XCI, 806.

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care so as to raise the temperature gradually, as otherwise the water vapor, passing off too quickly, will open joints of the masonry through which later on the lead in the crucible will percolate.

It takes 24 hr. or more to dry and warm the brickwork of the crucible. There are various ways of doing this. The following procedure is satisfactory. The water is turned into the jackets so as to fill them and have a slight overflow; the damper in the down-take is partly closed, the side- or end-charging doors are closed with furnaces having a high top, or the charging-opening on the feed-floor is covered with an iron plate, if the gases are withdrawn beneath it. A wood fire is made on the bottom of the furnace that will not reach halfway up the crucible. If it is kept going for a few hours, always replenishing the wood, ashes will have collected in the crucible. These, being bad conductors of heat, have to be removed in order that the burning wood may be in contact with the bottom. When these are raked out by means of a hoe from the breast of the furnace, a new fire is kindled. After from 3 to 4 hr. too many ashes will have accumulated in the furnace for the heat to have the desired effect, and the crucible is cleaned out again. While the crucible is being dried and warmed, the lead-well is filled with glowing charcoal, and the basin itself covered by a piece of sheet iron, so as to admit only a little air, thus preventing the charcoal from being burned quickly. Similarly, the breast of the furnace is closed with loosely set bricks, by which the draught is checked and too quick combustion of fuel on the surface prevented. The heating is continued for 24 hr., when the outside of the crucible will feel warm to the touch. This shows that all the moisture is expelled and that the crucible can stand a high heat without endangering the brickwork.

The next step is governed by the method used for filling the crucible with lead. It is essential for good work to have a clean crucible entirely filled with red-hot lead. If filled in part with charcoal ashes and small bits of charcoal, a dead layer will be formed between lead and slag which soon will harden to a crust and attract small obstacles that would otherwise be carried by the matte and slag.

After supplying the crucible with the necessary lead, the first charges[1] will consist of an easy-smelting slag with much coke and the fluxes necessary to slag the coke-ash. Ore-charges gradually replace slag-charges, and the high percentage of fuel is diminished until finally the normal charge is reached. It is better for a blowing-in slag to be glassy than crystalline, as it melts more readily, hence a chilled slag is preferable to one that has cooled slowly. The slag consisting of SiO_2 30, FeO 40, CaO 20 is better than any of the other types given in Table 64.

There are three ways of supplying the necessary lead to the furnace, by the lead-melting method with its variations, the platform method, and the lead-charging method.

162. Lead-melting Method.—This presupposes that the crucible has been brought to a red-heat before the lead is introduced. In order to accomplish

[1] Hofman, "General Metallurgy," 1918, p. 487.

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this, a wood fire is kindled in the warmed crucible and some charcoal added; the blower is started and made to run slowly; the bags of the tuyères are tied or wound up, or the blast-gates are closed, excepting that of a tuyère nearest the breast, which is connected with an iron pipe, and the latter inserted into the crucible from the breast. The blast is allowed to play on to the charcoal until it is well ablaze. A second layer of charcoal is added and, when fully aglow, a third, and so on until the crucible is filled to the jackets. Thereupon the pipe is then pushed down deeply into the glowing coal in order that the blast may reach the bottom of the crucible. Meanwhile the furnace-man works at intervals with a rod and a hoe, turns over the coal, and moves it front to back and *vice versa*, so as to get it all into a perfect glow. When this has been accomplished, the pipe, of which often a small part has been burnt off, is withdrawn, and the furnace let alone for an hour or two. The pipe is again introduced, and the charcoal burnt down, the furnace-man stirring it to bring all parts into contact with the blast. The ashes are removed, and the crucible and lead-well thoroughly cleaned. This heating is repeated from three to four times in 24 hr. The outside of the crucible will then have become too hot to be touched with the hand.

Up to about 20 years ago it was customary to fill the crucible again with glowing charcoal as outlined, to feed charcoal from the throat to reach about 1 ft. above the tuyères, to add a bed of coke extending to the top of the jackets, to charge lead and coke with some slag and the necessary fluxes, and to fill the furnace with alternate layers of half-ore charge and half-slag charge, using the amount of fuel necessary for the full-ore charge. Then followed, uncovering of lead-well, clearing of tuyère-holes, insertion of tuyère-pipes, and starting of blower, having it make as few revolutions as possible and gradually increasing its speed.

Another method, very common, was to put in the breast, close the tuyère-openings, cover the lead-well, feed from the throat charcoal to the top of the jackets, and cover the charcoal with a bed of coke about 1 ft. thick; then have a man descend into the furnace and fill it in part with blowing-in charges, consisting of lead, coke, slag and fluxes, and end with regular ore-charges. When the furnace had been thus filled, the charcoal was kindled from the tuyères, the pipes were put in place, and the blower was started and made to run slowly.

Both methods customary with European furnaces tapped at the bottom of the crucible and adapted to American furnaces having an Arents siphon-tap, have been given up, as even with good charcoal there are likely to be formed blowing-in crusts which, at best, are difficult to remove, and frequently remain in the crucible during an entire campaign, interfering with the normal working. With the perverse tendency of a crust to grow during a campaign, there is danger of the crucible freezing up solid. The use of coke instead of charcoal only makes things worse.

This led to the present lead-melting method. The crucible and well, heated as shown, are cleaned, the well is filled with glowing charcoal and covered, a fire is started in the crucible, and, when this is filled in part with glowing coals,

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bars of lead are introduced from the breast by sliding them in on a paddle. As the lead melts, charging is continued until the crucible is filled with molten lead; the fire floating on the lead is kept going by using blast intermittently. When the crucible has been thus filled, the last embers and ashes are raked out, enough dry kindling is charged from the front and then from the feed-floor to reach well above the jackets, and then the breast is put in. On top of this comes a 2-ft. bed of coke, to which bars of lead are added in order that this, coming down hot, may help to heat the lead in the crucible. Then follow the usual slag- and ore-charges. The wood is kindled through the tuyère-openings, the tuyère-pipes are connected, the blower is started, and a gentle blast maintained. The feeding of bars of lead from the top is often continued until the lead in the crucible has become red-hot.

At some works, having in operation several blast furnaces and a drossing kettle, lead is taken from the kettle and poured into the crucible instead of being melted down in it. Thus, *e.g.*, with a furnace 48 by 160 in. at tuyères, there are charged on top of the liquid lead, 3 ft. dry cedar kindling, a bed of coke 1 ft. deep, 6 slag-charges consisting of 8000 lb. slag-shells, 800 lb. iron ore (Fe 56 per cent.) and 1000 lb. coke, and then slag-ore charges, made up of 8000 lb. ore and 2000 lb. slag, with 13 per cent. coke. These are run for 2 days, whereafter slag is omitted, and the full ore-charge of 10,000 lb. with 12 per cent. coke is fed. Care is taken that the kindling is uniformly ignited with oil-waste. During the first 24 hr. the blast-pressure is not allowed to exceed 24 oz. per sq. in.; on the second day it is slightly raised, and this increase continued until at the fifth or sixth day it has attained its normal pressure of from 38 to 42 oz. per sq. in. When the blast has been started, the first slag appears after several hours, and matte is noticed in the slag about 18 to 24 hr. later.

In another instance, with a furnace 42 by 192 in. at tuyères, there is charged on to the liquid lead dry kindling to reach well above the tuyères. Then follow, 3000 lb. coke (forming a bed 3 ft. deep) and 4000 lb. slag; 3000 lb. coke and 3000 lb. slag with fluxes to slag coke-ash; and 825 lb. coke, 6000 lb. charge, with additional 1200 and 1500 lb. slag fed along the sides. The furnace is filled with the last charge to 4 or 5 ft. from the throat, the kindling is ignited, and enough blast put on to show a pressure of about 20 oz. per sq. in. In about 2 hr. after starting the blast, the first slag appears at the tuyères. When the first matte is seen in the slag, the pressure of the blast is gradually raised to 36 oz. per sq. in.

It may be added that many metallurgists add matte to their first ore-charges, as this comes down hot and has a decided cleaning effect, since it takes up materials which the slag may refuse to carry out.

The lead-method of starting requires bringing the crucible to a red heat; and this calls for much time, labor, and fuel; care must be taken that the kindling placed on the lead shall not ignite ahead of time; there is danger of uneven distribution of the charges in the necessary quick filling of the furnace; there is likelihood of all the kindling not being completely burnt if it has been pressed

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down into the lead, with the result of wood more or less charred floating on the lead and forming the nucleus of a blowing-in crust. In spite of these disadvantages the method has given and is giving satisfactory results.

163. Platform Method.—This method was devised to do away with the necessity of melting lead in the crucible, and to permit an even distribution of the charges. The crucible is dried and warmed in from 2 to 3 days; a rough platform, Fig. 326, of 1½ by 3-in. lumber, well soaked in water, is built over the crucible above the slag-tap and beneath the tuyère-holes, leaving spaces 1½ in. wide between the planks, through which later on the lead may find its way into the crucible. In front of each tuyère is placed finely split kindling, and on the platform cord wood to a depth of from 3 to 5 ft.; on top of this come lead, coke, and blowing-in charges. With a furnace 12 ft. 9 in. by 3 ft. 6 in. at tuyères and a crucible 2 ft. 6 in. deep, tapering from 3 ft. 6 in. to 2 ft., there are placed on the platform 4 cords of wood, and on top from 35,000 to 40,000 lb. of pig lead with an old crucible, or more with a new. Then come the blowing-in charges given in Table 79. The blowing-in charges are followed by the regular ore-charges of 8600 lb. with an extra 100 lb. coke for the first 24 hr. When the furnace is filled, the kindling is ignited with burning oil-soaked waste, which is pushed in through the tuyère-openings, and a gentle blast turned on; the volume of air is then gradually increased in the usual way. The wood and coke melt the lead, which trickles down and fills the crucible. The slag is supposed to melt only after the crucible has been filled with molten lead. When it has risen to the tuyères, it is tapped, and the volume of air further increased. Many furnaces have been blown in satisfactorily with this method. In case, however, the platform is too weak and breaks on account of the sudden settling of the charge, or if it burns prematurely, part of the charge may drop into the crucible. There is then little chance of its being floated completely by the lead, and the crucible will

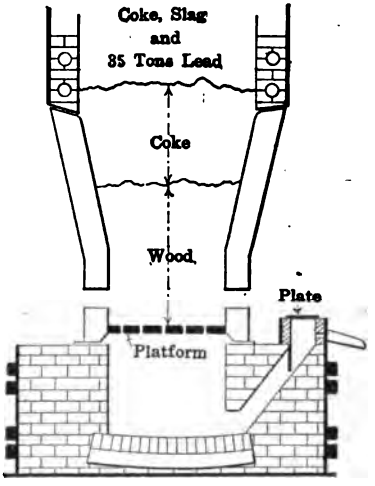

FIG. 326.—Platform for blowing in blast furnace.

TABLE 79.—BLOWING-IN CHARGES

Number of charges	Cars coke @ 650 lb.	Cars slag @ 1,200 lb.
1	6	.....
1	3	5
2	2	6
1	3	5
1	2	6

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remain partly filled with foreign matter, which interferes with regular work and is very likely to cause freezing of the crucible contents. Again, if the planks are completely burnt, partly-charred wood will float on the lead and form a nucleus for accretions. Very few, if any, furnaces are blown in at present in the manner described.

164. Lead-charging Method.—In this method, which originated at Mapimi, Mexico, bars of lead are piled in the crucible as shown in Fig. 327; they are followed by wood, lead, and blowing-in charges. In the example to be given, the crucible is 46 by 162 in., and holds about 65,000 lb. of pig lead. The crucible, having been warmed for 48 hr., is cleaned; from 50,000 to 55,000 lb. of lead in 100-lb. pigs is stacked with 1-in. spaces as shown, and reach to from 3 to 4 in. beneath the tuyère-level. Kindling and oil-soaked waste are placed in front of the tuyère-openings; front and back of furnace are closed; cord wood is charged from the throat to a depth of about 3 ft., being introduced from both sides of the furnace in order that it may be distributed evenly; then follow the blowing-in charges given in Table 80. The blowing-in charges are followed by the regular ore-charges, to which some raw matte is added to wash out the tuyère-region; during the first 24 hr. an extra barrow of coke is added to every fourth charge.

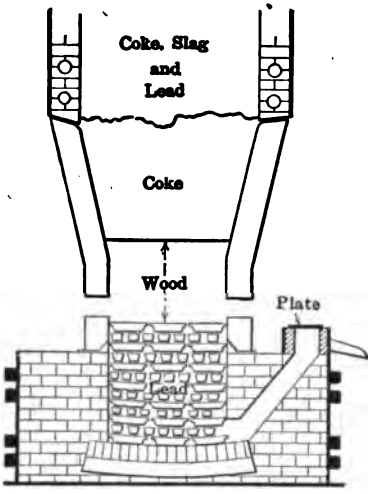

FIG. 327.—Lead-charging for blowing in blast furnace.

When the furnace is filled, the lead-well is covered with an iron plate which is luted, and weighted with a bar of lead. The oil-waste and kindling at the tuyère-pipes are ignited and allowed to burn for 5 to 10 min. The tuyères are put

TABLE 80.—BLOWING-IN CHARGES

Charge No.	Barrows coke (80 p.c.F.C.) @ 85 kg.	Buggies slag @ 850 kg.	Barrows matte @ 150 kg.	Bars lead @ 50 kg.	Ore-charges @ 850–1,000 kg.
1	18	.....	.....	.....	.....
2	8	4	.....	.....	.....
3	8	4	.....	.....	.....
4	8	4	.....	50	.....
5	8	8	.....	.....	.....
6	8	8	.....	.....	.....
7	8	8	1	.....	.....
8	8	8	1	.....	.....
9	6	4	.....	.....	2
10	6	4	.....	.....	2
11	6	4	.....	.....	2

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in, and the blower is started to give a pressure of ½ oz. per sq. in.; the volume of air is increased during the next 2 hr. to furnish a blast-pressure of from 7 to 8 oz. The bottom of the lead-well becomes sealed with lead in 1 to 1½ hr. after starting; the cover is removed, and the lead melted and kept liquid with an oil-burner. Two hours after starting, the first slag appears at the tuyères; this is allowed to rise to the tuyères and tapped. The tap-hole is worked with a rod to remove ashes; the blast is allowed to blow through to assist in removing them. The blast volume is raised during the next 4 to 5 hr. to furnish a pressure of about 16 oz. If the lead in the well has not been liquefied by use of an oil-burner, the solid plug of lead will melt in about 8 hr. after putting on the blast, and hot lead will rise quickly in the well. In case the lead does remain solid, a red-hot steel bar is driven through the plug. Between the eighth and twenty-fourth hour after starting, the blast is raised to furnish 26 oz., and during the next 12 hr. 36 oz. pressure. The rate of increasing the blast is governed by the surface of the charge, which should remain cool and sink evenly. During the first 36 hr. after starting, the top of the charge is kept from 4 to 5 ft. down from the feed-floor. In 36 to 48 hr. after starting, when the extra coke of the blowing-in charges has been burnt, the blast is raised to furnish 40 to 42 oz., and after 72 hr. the normal 48 oz. pressure.

This method is usually satisfactory, and is being adopted by many smelteries. At Mapimi, where there is a speiss-fall of from 2.5 to 4.0 per cent., blowing-in crusts or sows are not formed.

165. Work on the Charging Floor.—The work on the charging floor consists in assembling ore, flux, and fuel; weighing the required amounts; preparing the charge; and feeding it into the furnace.

The older as well as the more recent methods of assembling ore, flux and fuel have been discussed in §114; hand-feeding has been treated in §109.

166. Mechanical Feeding.[1]—The charge components, made up of pieces of ore, flux, and fuel, vary much in size. A coarse charge is preferred to one that is fine, as the ascent of gases is more uniform; but, if too coarse, the gases pass off too quickly and do not prepare the charge to the required degree. A fine charge offers too much obstruction to the upward travel of the gases; these break through at different points (form blow-holes), do not give the charge the necessary thermal and chemical preparation, and cause loss by carrying away dust and fume. There is also danger of fines trickling through the coarser parts of the charge and arriving in a crude state at the smelting zone; they will chill the furnace, and may even fill the tuyères. This may be obviated by the Glenn filter charge[2] which consists in making the charge large and feeding upon a bed of coke first the coarser parts of ore and flux, then the finer, and lastly the fines. At present, when charges are made up largely of blast roasted mate-

[1] Dwight, *Tr. A. I. M. E.*, 1902, XXXII, 380.
Hixon, *Eng. Min. J.*, 1903, LXXV, 267.
Hofman, *Min. Ind.*, 1903, XII, 260.
Anderson, *Eng. Min. J.*, 1916, CI, 885.

[2] *Eng. Min. J.*, 1884, XXXVIII, 34.

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rials, the amount of fines to be treated has been reduced to such a degree that they do not cause any trouble. A correct mixture of coarse and fine is therefore desired. Of the ore,[1] one-third might be in pieces from 5 to 2 in. in diameter, one-third from 2 to ½ in., and the remaining third from ½ in. down. Flux ought to be about of the same size as ore; foul slag not larger than 6 in.; and coke ought to be broken somewhat. As long as the amount of coke-fines is not excessive (see §124), there is no reason for separating them from the rest by forking.

Wetting a charge while bedding or before feeding into the furnace is advantageous with an excess of fines over the normal, as it prevents unmixing when the charge is being handled.

It has been shown in §109 that in hand-feeding a charge, the coarse ought to be placed near the center, and the fine near the side. The coke is also distributed more along the center than the side, especially as, being lighter than the rest of the charge, it is likely to be pushed toward the walls in its descent through the furnace. Guyard[2] advocates charging fuel alternately toward the center and sides to obtain an even ascent of gases; he believes that feeding of fuel toward the center and charge toward the sides, causes wall accretions to form immediately above the jackets, whereas alternate feeding of fuel toward center and sides will cause accretions to form higher up in the furnace.

A strict adherence to the principle of placing coarse toward the center would draw the gases too much toward the center, and cause premature slagging and overfire, and leave the finer charge along the sides imperfectly prepared. What is wanted is that the gas current ascend uniformly over the whole area of the smelting column, and this usually is the case when the smoke passes off uniformly and quietly from the surface of the charge, if it is cool and sinks regularly and evenly.

A mechanical feeding apparatus, to be successful, must fulfill the requirements given as regards distribution of charge. Omitting the earlier trials mentioned in §109, the first mechanical feed which had any marked success was the so-called Pueblo system, discussed in §167.

167. Pueblo System.[3]—This was put into operation at Pueblo, Colo., about 1895. Though it is ancient history as far as present practice is concerned, it embodies features of sufficient permanent value to warrant noting some of the details. Fig. 328 gives a vertical longitudinal section through the inclined hoist. In the center is the charging pit with charging car placed crosswise; to the left on the ground floor are the ore-bedding bins, and on the second floor is the sampling-building; to the right are the bins for fluxes and fuel, and adjoining, a battery of seven blast-furnaces, 60 by 120 in. at tuyères with a working height of 20 ft. The tuyères, six on a side, are 4 in. in diameter; they have Mathewson water-cooled nozzles (Figs. 186 and 187) projecting 6 in. inside the jackets, thus

[1] Dwight, *loc. cit.*

[2] Emmons, "Geology and Mining Industry of Leadville," U. S. Geol. Surv., Monograph XII, p. 665.

[3] Dwight, *Tr. A. I. M. E.*, 1902, XXXII, 375.

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leaving a smelting distance of 48 in. between them. The furnace-tops are closed by hinged and counter-weighted heavy sheet-iron plates; the furnace gases are withdrawn below the charging floor. The side-walls have iron door-frames, bricked up while the furnace is in operation, in order to permit repairing or barring off wall-accretions without interfering with the charging apparatus.

Ores arrive on the elevated track to the left of the sampling-mill and are sampled by fractional selection. The samples are discharged to the right on to an elevated platform in the sampling-mill and held in separate stalls until they are passed either through crushers or chutes on to the main floor, where they are cut down by hand, usually by quartering. The rejected ores are unloaded to the left into V-shaped bins, whence they are discharged into Hunt automatic dump cars[1] and distributed over the ore-bedding bins each holding about 2000 tons of ore. The bottom of a Hunt car has the form of an inverted V sloping toward the inclined sides, which swing open, when released by a lever, and allow the ore to be discharged on either side of the inclined track running over the center of the bin. The car is attached by a cable to a counterpoise which is heavier than the car when empty and lighter than when it is filled with ore. Thus the car will run down the incline when filled and up again to its original position when emptied. The unloading is effected automatically by an adjustable stop along the side of the track, which strikes the lever and thus releases the sides. By changing the position of the stop, the car can be emptied at any point along the center line of a bedding-bin, and the ore is then spread out by hand. As one man can attend to several cars, the labor involved in bedding the ore is reduced to an extremely low figure.

Fluxes and fuel arrive on the elevated track to the right of the pit. The samples,

[1] Sadtler, *Colo. School Min. Quart.*, 1893, II, 33.

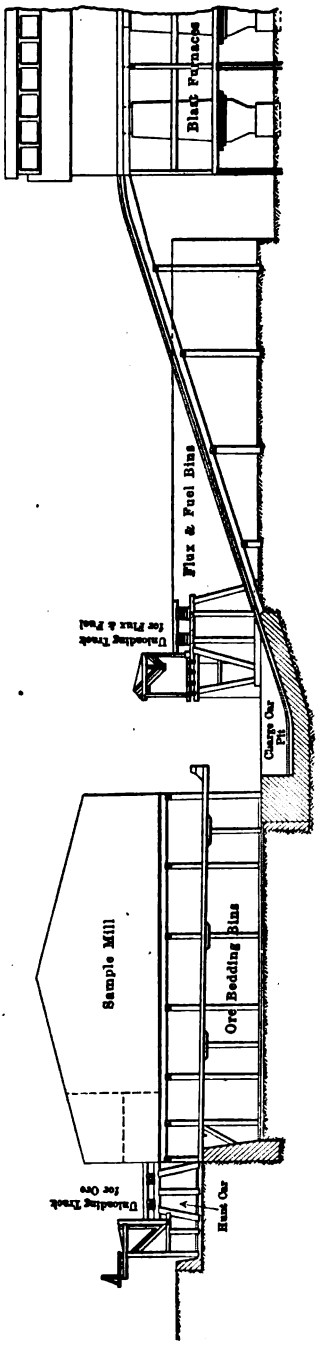

FIG. 328.—Pueblo system of feeding the blast furnace.

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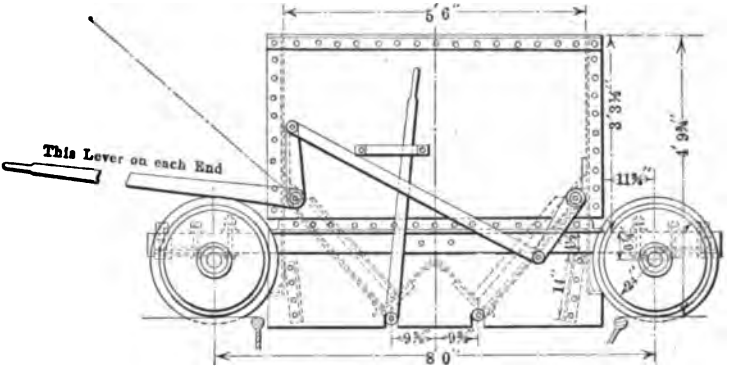

FIG. 329.—End elevation.

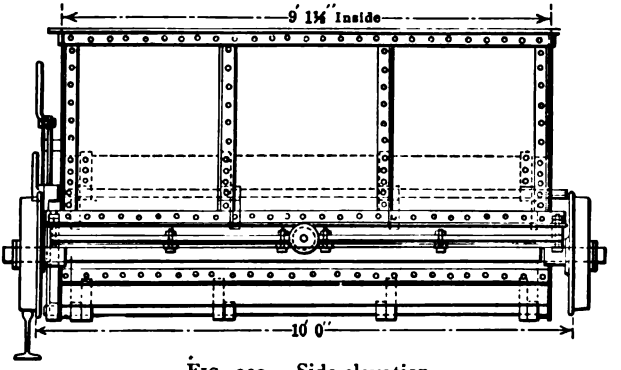

FIG. 330.—Side elevation.

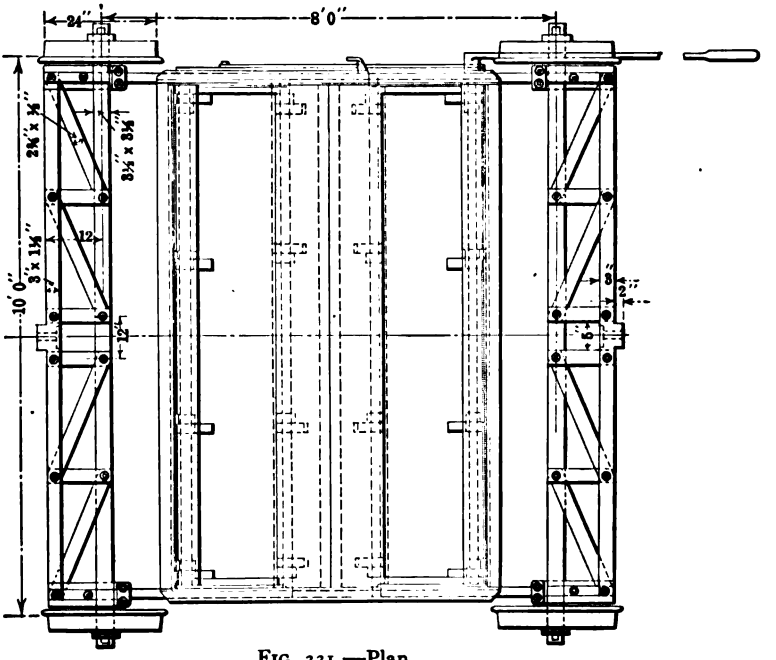

FIG. 331.—Plan.

FIGS. 329 to 331.—Williams charging car.

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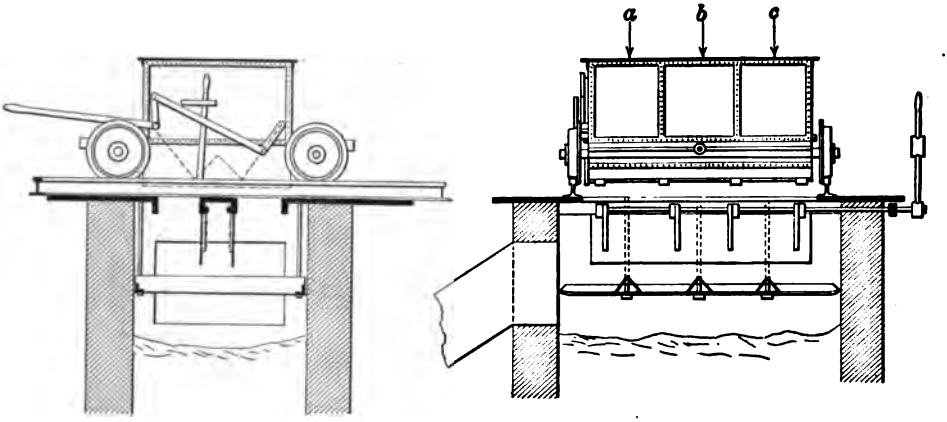
when taken, are collected in V-shaped bins near the track and discharged into a car running to the sampling-mill. The rejected materials go into the flux and fuel bins.

Figs. 329–331 give side and end elevations and plan of the Williams charging car, which holds about 5 tons charge. It is 5 ft. 6 in. by 9 ft. 1½ in., has vertical sides to a depth of 3 ft. 3½ in., an A-shaped bottom, and hinged discharge-doors to be opened and closed by levers at each end. The car is filled at the ends from buggies by wheelers who work in pairs. Each wheeler weighs into his buggy one-half charge of a particular component, takes his place in the line of wheelers on either side of the charging-pit, and discharges his buggy into the opposite ends of the car, partners discharging equal quantities simultaneously, until the charge is complete. Coke is added last to avoid breaking.



FIGS. 332 and 333.—Sections through furnace-top.

The filled car is hauled by a steel tail-rope up the inclined trestle-track on an angle of 17° to the feed-floor, and stopped at any furnace that is to be fed; the gauge of the track is about the same as the length of the throat of a furnace. The hoisting drums are situated within the furnace building at the crest of the incline; at the farther end of the building is a tightener-sheave to keep the tail-rope taut.

The charge in the car will show two conical piles near the ends with apices of fines under the letters *a* and *c*, Figs. 332 and 333, and coarse parts in the valley, *b*, between the piles. This irregular distribution of coarse and fine is corrected by three A-shaped cast-iron deflectors, extending across the shaft about 18 in. above the stock-line, which divide the shaft into four rectangular areas; they scatter the coarse and fine materials as these drop from the car, and make a more uniform mixture.

When the car has been spotted over a furnace, the hinged doors of the furnace are dropped, as shown in Figs. 332 and 333, and the discharge-doors of the car are opened. The charge falling strikes the A-shaped bottom of the car, is thrown against the walls, whereupon the deflectors correct uneven distribu-

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tion of coarse and fine, at least to some extent. It will be noticed that this arrangement necessitates a drop of charge of about 10 ft.; this causes breaking of coke and packing of charge-components, both undesirable features.

At Pueblo there was effected a saving of 9 cts. per ton with a daily smelting

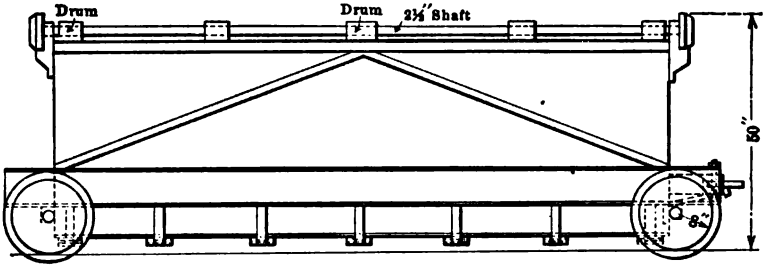

FIG. 334.—Side elevation.

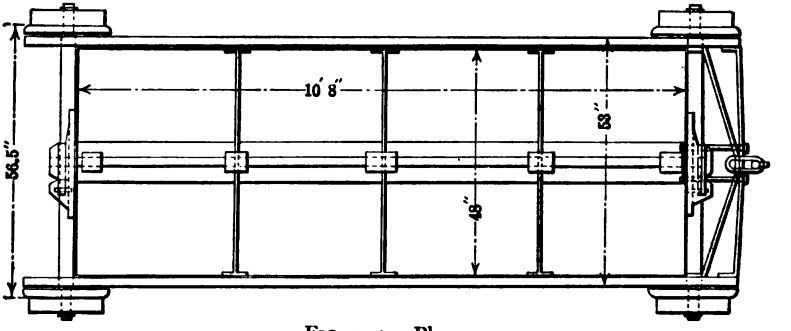

FIG. 335.—Plan.

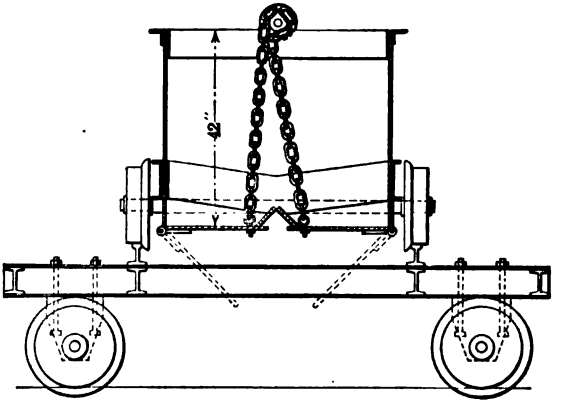

FIG. 336.—End elevation and transfer car.

FIGS. 334 to 336.—East Helena charge-car.

capacity of 700 tons charge. With decrease in size of plant this saving falls off quickly, so that soon the advantage disappears entirely.

168. East Helena System.[1]—This method of mechanical feeding developed

[1] Dwight, *Tr. A. I. M. E.*, 1902, XXXII, 380.

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by Dwight in 1900 at East Helena, Mont., is at present in use at the great majority of lead plants. Its original form has undergone some modifications.

The manner of assembling charge-components at East Helena is similar to the one shown in Figs. 220 and 221; that is, the charge-car is hoisted up the incline endwise, and delivered to a transfer carriage which is moved by a tail-rope system over the tops of a battery of furnaces from which the gases are withdrawn beneath the feed-floor.

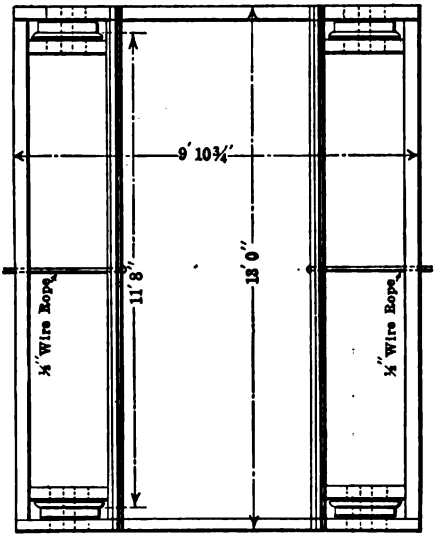

FIG. 337.—Plan, East Helena transfer-car.

The East Helena charge-car is shown in Figs. 334–336, and the transfer-car in Fig. 337. In Fig. 336 the charge-car stands on the transfer-car. The charge-car is 10 by 4 ft. and 3 ft. 6 in. high; it holds 8800 lb. of ore and flux and 3200 lb. of coke, or 6 tons in all. The bottom consists of a fixed A-shaped distributor and two doors hinged at the sides, which are held in place by means of chains wound on a longitudinal windlass-shaft on top of the car. The transfer-car is 13 by 9 ft. 10¾ in., travels on track of 11 ft. 8 in. gauge, and is hauled by a 1½-in. wire rope.

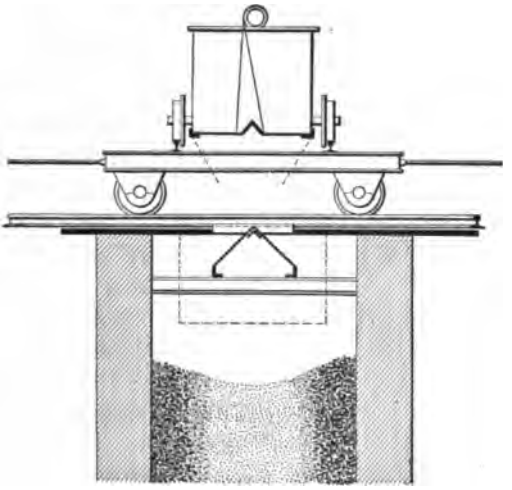

FIG. 338.—East Helena feed-car and Dwight spreader.

When mechanical charging was installed by Hixon,[1] the contents of the car were dropped through the central feeding-slot in the iron plates forming the furnace top as shown in Fig. 190.. The result was that the fines accumulated along the center, while the coarse rolled toward the sides; the charge was wrongly distributed, and the furnace showed overfire. Dwight[2] corrected this by placing an A-shaped spreader beneath the feed-slot as shown in Fig. 338; the charge falling from the car on to the spreader is thrown toward the sides where most of the fines remain, while the

[1] *Loc. cit.*
[2] *Loc. cit.*

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coarser parts roll down the inclined surface of the descending charge and gather along the center.

In many instances this separation of coarse and fine was carried too far, as the heat crept up along the center of the furnace, and the ore along the sides was insufficiently prepared. This was corrected by W. W. Norton as shown in Fig. 339*a*. The A-shaped spreader, instead of being a single casting; Fig. 339, is made up of several parts separated by open spaces, Fig. 339*a*. The charge, on dropping from the car, is not deflected altogether toward the walls, but one part falls through the openings between the channels and lands directly in the center. There is thus obtained a satisfactory distribution of charge, much fine with some coarse along the sides, and much coarse with some fine along the center.

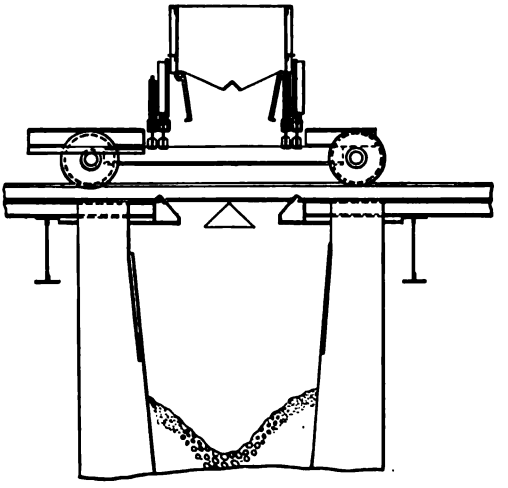

FIG. 339.

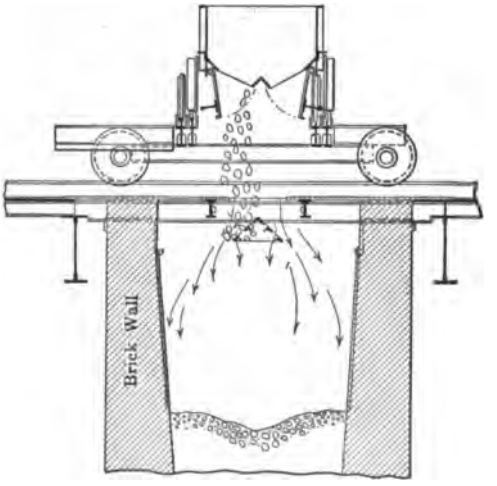

FIG. 339*a*.

FIGS. 339 and 339*a*.—Dwight-Norton spreader.

In using a transfer-car, the charge has to drop a considerable distance. The height of fall may be reduced to as little as 2 ft. if the transfer-car is omitted. This is the case with the blast furnaces of the St. Joseph Lead Co. at Herculaneum, Mo. The standard furnace is 42 by 192 in. at tuyères and 72 by 192 in. at throat, has a working height of 12 ft. 4 in., and four boshed jackets on a side, each with three 3-in. tuyères. The shaft is of brickwork; the inner sides are protected by cast-iron plates, which extend from the throat to the surface of the charge; further details are given in Table 53. The throat of the furnace, Fig. 340, is covered with cast-iron plates resting on rollers so that they can be easily removed to one side for barring down. The charging-car, 14 ft. 7 in. by 3 ft. 7 in. and 4 ft. 2¾ in. high, has hinged discharge-doors which are balanced by weights and manipulated by levers. It travels on rails carried by I-beams. The spreader is made up of cast-iron

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angles with intervening spaces. The center angle, as shown in Fig. 340, has been removed in order to charge more fine material toward the middle of the furnace. As the car is not as long as the throat of the furnace, the spreader has been shortened so that it extends only to within 18 in. of the ends of the furnace. With this change, the charge-car showed a tendency to build at the back; a deflecting chute placed toward the front at an angle of about 60° corrected this evil in that more charge is delivered toward the front than the back. The height through which the charge has to drop, the distance from bottom of car to top of downcomer, is 6 ft. $3\frac{1}{2}$ in.

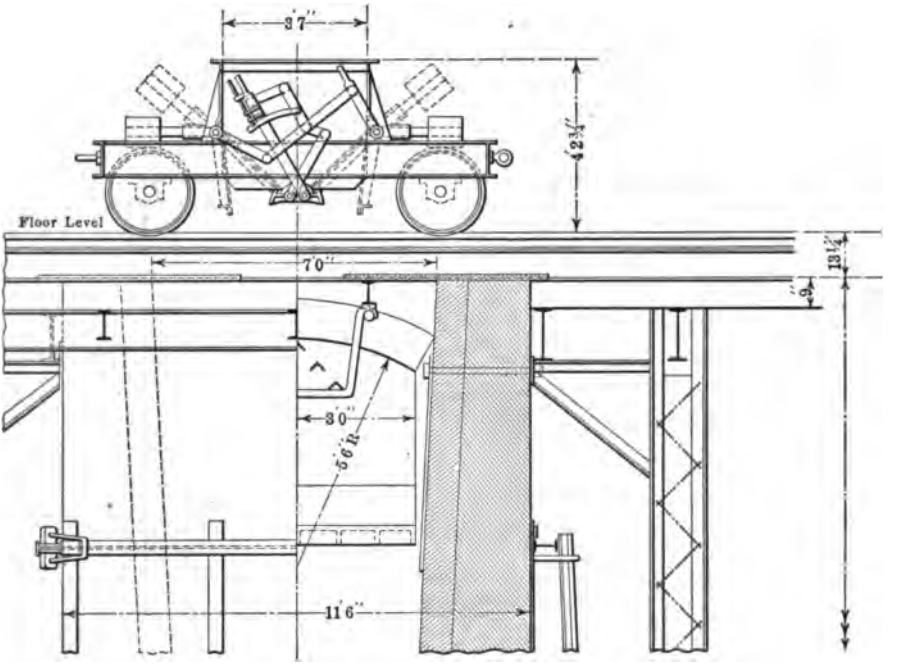

FIG. 340.—Furnace-top, St. Joseph Lead Co., Herculaneum, Mo.

Details of the spreader in use at works of the United States Smelting Co. at Midvale, Utah,[1] are shown in Fig. 341. The preparation and feeding of charges have been taken up in §114, and blast furnace details in Table 53. The spreader, suspended from chains, has small spaces between the angles, and the central part is left open so that a considerable part of the charge is delivered along the median line of the furnace without any distribution of coarse and fine.

With charges made up largely of blast roasted material, and the resulting greater smelting power of the furnace, there is less danger of the heat creeping up in the furnace; this has resulted in an endeavor to distribute coarse and fine more evenly than was formerly the case.

Some of t he furnaces of the Consolidated Mining and Smelting Co. of Canada,

[1] Anderson, *Eng. Min. J.*, 1916, CI, 885.

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Trail, B. C., as shown in Figs. 176–179, have open tops, the gases being withdrawn from the sides. The feed-opening of a furnace is closed by a plate running on wheels which travel on the same track as the charge-car. The latter, arriving in front of a dinky locomotive, pushes ahead the plate, delivers its charge in the usual way on to the angle-spreader, and pulls back the plate into place when it returns to receive another charge. By keeping the feeding-slot closed, the volume of gases to be treated in the Cottrell precipitator is much reduced.

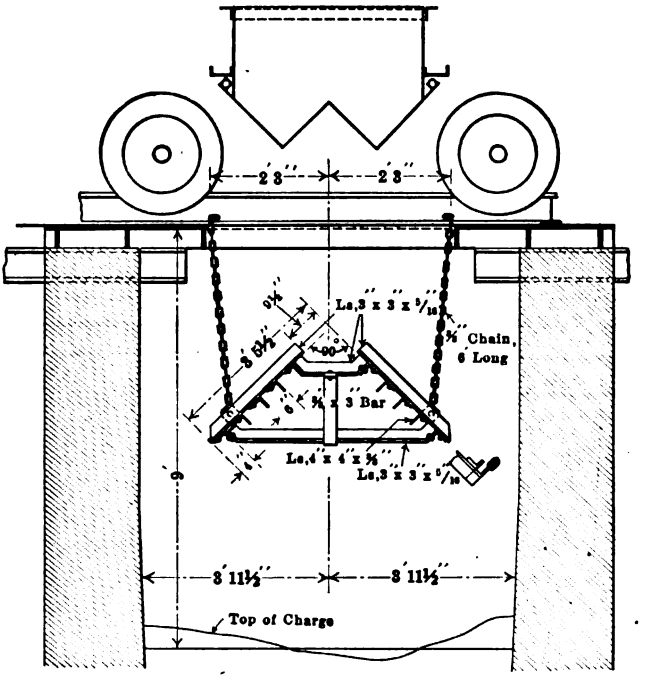

FIG. 341.—Charge-spreader, Midvale, Utah.

This arrangement recalls the Freeland charging machine[1] in operation at the copper blast furnaces of Ducktown, Tenn.

Other lead furnaces are fed in a manner similar to that in operation at Northport, Wash. (see below); a third set is fed by cars with side-wheels similar to those in use at the blast furnaces of Grand Forks, B. C.[2]

The furnaces of the Northport Smelting and Refining Co.; shown in Figs. 180–183, are hooded; *i.e.*, a furnace has a brick top from which the gases pass off through a downcomer into a dust chamber, and the charges are fed from the ends. The tracks, of 30-in. gauge, on the feed-floor enter the furnace at one end and extend to the other. The charge-car is pushed by a dinky locomo-

[1] U. S. Patent, No. 768596, Aug. 30, 1904.
Renwick, *Min. Sc. Press*, 1913, CVI, 443.
Browne, *Bull. Canad. Min. Instit.*, November, 1916, p. 976.
[2] "Hofman, "Copper," 1918, p. 166.

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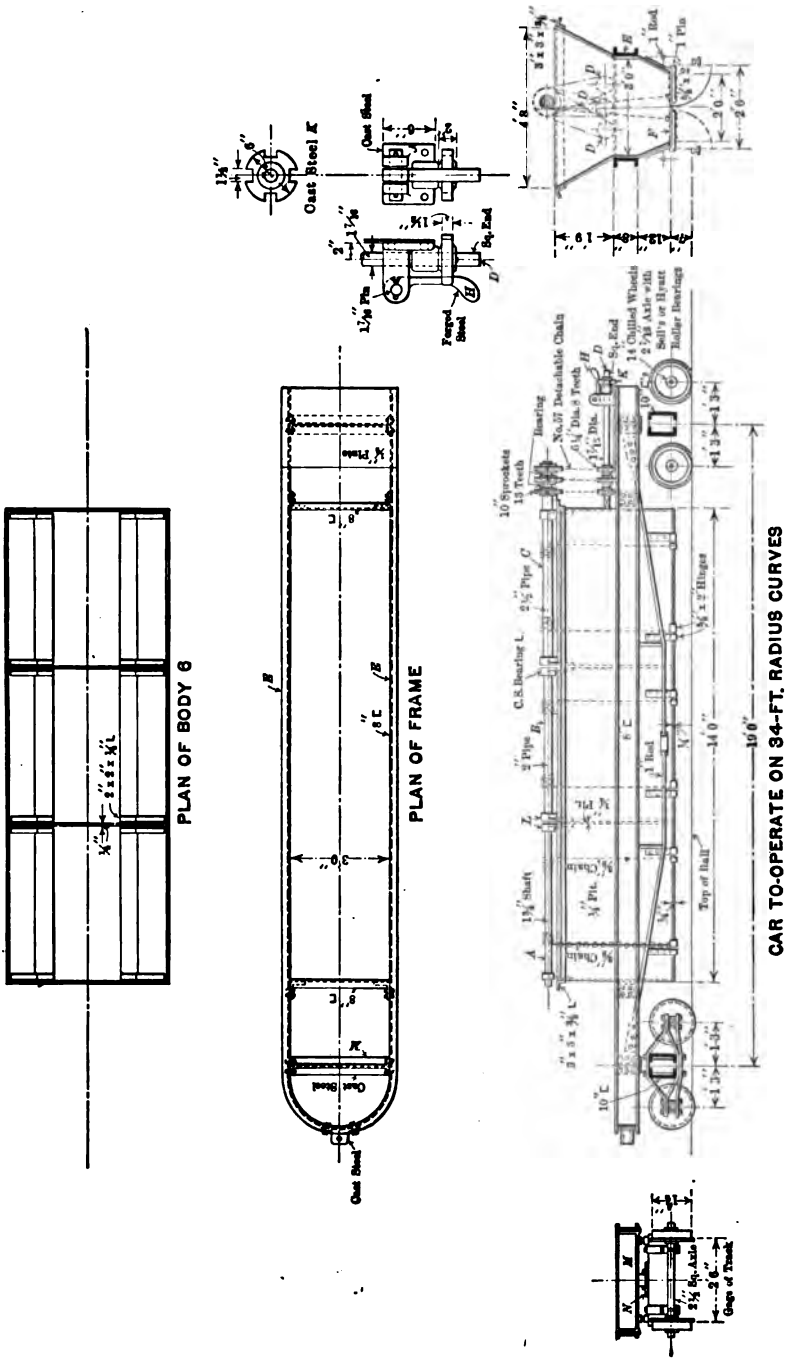

FIGS. 342 to 348.—Charge-car, Northport Smelting & Refining Co., Northport, Wash.

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tive into the furnace and discharged on to a spreader. This consists of a 6 by 6 in. angle running along the center of the furnace about 18 in. below the feed-floor. The side-walls are protected by 6 by 6-in. angles. The charge-car, shown in Figs. 342–348, holds 7000 lb. charge and coke. It is 14 ft. long, 4 ft. 8 in. wide at the top, 3 ft. in the middle, and narrows to 2 ft. at the discharge; the depth is 3 ft. 5 in. The flat bottom is made of three pairs of hinged doors, which close the three compartments of the car. The discharge-doors are attached at the free ends to ¾-in. chains wound severally around a longitudinal 1¾-in. solid shaft, enclosed in a 2-in., and this in a 2½-in. pipe-shaft. The shaftings carry at one end 10-in. sprocket-wheels connected by chains to gears which are severally rotated by hand through a clutch by a main shaft, 1 7/10 in. in diameter and square at one end. The three compartments give latitude in meeting the demands of a furnace when the charges do not sink evenly.

169. Darby Thimble Method.—The details of this feeding system at the Selby Lead Works and the smeltery of the Sulphide Corporation have been taken up in §109.

170. Work on the Furnace-floor.—This consists mainly in regulating the water-supply, taking care of the tuyères, and disposing of the molten products.

The water of the jackets is kept at about 70° C., the usual test being that the hand can be quickly passed through the outflowing water without being scalded. Any irregularity in the temperature of the jackets indicates that the coating on the inside is thicker on some than on others, and thereby that the smelting proceeds unevenly; the slower descent of the charges on the cold side will corroborate this on the feed-floor.

The pressure of the blast is watched, and all the changes in the gauge are noted. It is regulated by means of a damper. If the furnace has its own blower, its revolutions are also counted. Self-recording pressure gauges are used in most plants.

The appearance of the tuyères forms a good indication of the inner condition of the furnace. It is not necessary that the entire mouth of the tuyère should be bright; it is usually covered by a thin scale of slag, showing a star-like brightness in different places. If the tuyère becomes quite dark, a bar is inserted and the slag pierced to see if it has grown too thick. In this case repeated poking only aggravates matters, as a "nose" of chilled slag will form which reaches into the furnace. (For correction of this evil see §172.)

Speiss, matte, and slag are usually tapped into a settler (§118) from which the slag overflows into a receiver to be hauled to the dump or granulated (§120), and speiss and matte are discharged periodically (§117).

The lead is usually tapped from the well (§115) into a pot and cast into bars; in the absence of a well, it is tapped with speiss and matte into a shallow basin, from which it is ladled into bars after the solidified cake of speiss and matte has been removed.

In tapping the slag, the furnace-man used to have two ¾-in. steel bars, 6 to 8 ft. long, and two 10-lb. double-faced sledges. With the increased size of furnaces the clay plug which closes the tap-hole rarely becomes sufficiently

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hard to require steel bars; two ½-in. rods, 12 to 16 ft. long, bent to a circular handle at one end usually serve for this purpose. With a rod of this length the furnace-man can stand a sufficient distance from the furnace in doing his work so as not to be exposed to the fumes arising from the tap-hole. There are provided two stopping-rods, of ½-in. iron and 12 to 16 ft. long, with disks 2½ in. in diameter for the clay plug, and two ¾-in. iron rods, also 12 to 16 ft. long, for rodding.

In tapping, it is the rule to keep some slag in the furnace. At certain intervals the lead and slag are allowed to rise in the furnace until the little blue flame, seen through the tuyères, indicates that the slag has nearly reached that level, when the furnace is tapped clean to see how much slag the furnace holds. If there is less slag than usual, it shows that the tuyère-section is not as free as it ought to be. It is advisable, especially with a long furnace, to allow the blast to blow on and off out of the tap-hole for a while to be sure that all slag has been removed. If this is not done, the slag is likely to build up around the tuyères, especially around those at the back; and this necessitates later on much rodding.

Matte, perhaps some speiss, and usually a small amount of lead are tapped from the fore-hearth with bars, of ¾-in. steel from 6 to 8 ft. long; the tap-hole is closed with a clay plug as is the slag-tap. The matte-tap usually is hard and requires sledging.

In order to make the tapping less arduous, the tapping-bar is driven by a few gentle strokes of the hammer into the soft-clay plug, with which the hole has been closed, until it reaches the hardened interior. The bar remains there until the next tapping, when only the inner crust has to be broken. An oxygen-torch to melt out the matte in the tap-hole has been successfully applied by Browne[1] and others;[2] the use of electricity for this purpose has been advocated by Thompson,[3] Hixon,[4] Welsh[5] and Veritas.[6]

The lead in the crucible is always at a lower level than in the well, as the pressure of the blast and the weight of slag and matte force it downward. It is essential that the lead in the furnace may rise to the same level as the slag-tap; if it does not, speiss and matte will be too far removed from the zone of fusion, and thus liable to cool and form a crust.

The less lead there is in the charge, the higher must be held its level in the crucible that it may not cool, as it is not frequently replaced by fresh lead that has just passed through the hottest part of the furnace. Formerly the tops of the crucible and the wells were on the same level; at present the well is built up to extend from 8 to 10 in. above the slag-tap.[7]

The lead used to be taken from the well by dipping and cast into bars (§115).

[1] *Tr. Canad. Min. Inst.*, 1915, XVIII, 91.
[2] McGill, Nev., *Eng. Min. J.*, 1917, CIV, 704.
[3] *Eng. Min. J.*, 1916, CI, 698.
[4] *Op. cit.*, p. 608.
[5] *Op. cit.*, p. 951.
[6] *Op. cit.*, 1916, CII, 517.
[7] von Schippenbach, *Metall. u. Erz.*, 1914, XI, 652.

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In doing this, there is danger of removing too much at a time; further, the bars are sure to contain much dross, which collects largely on the surface of a bar but is also irregularly disseminated through it. As the dross runs lower in Ag than the clean lead, much difficulty is encountered in sampling and later in assaying. At present the lead is always tapped at intervals from the well into a collecting pot, sometimes heated from a fireplace; the dross is skimmed and returned to the blast furnace, and the clean lead molded. With furnaces running charges high in lead, the collecting pot is hauled to a drossing-kettle and emptied, or the lead is run from the well into a casting machine, and the bars are then melted in a drossing-kettle.

In tapping from a furnace about 1000 lb. of lead, the charge will sink suddenly to occupy the space set free by the lead. This may result in particles of unmelted charge passing too quickly from the smelting zone to be fused, and forming the nucleus of a crust. For this reason some furnace-men prefer dipping to tapping, especially with low-lead charges; the amount to be dipped is then regulated by the number of charges that are fed into the furnace, and the percentage of lead they contain.

The clean bars of blast furnace lead are marked with a running lot-number; removed from the mold with a pick or otherwise; piled to be sampled, if this has not been done during the molding; weighed; and shipped.

A furnace is in good working order below, when the temperature of the jackets is uniformly high, the pressure of the blast does not fluctuate, and the tuyères remain bright, having only short noses. The furnace, within a given time, should produce the same amount of slag; the tap-hole should be neither too hard nor too soft; and the lead in the well should be of a bright red color, play with the blast, and sink slightly every time that slag is tapped.

On the furnace floor are required for every furnace, one furnace-keeper, one tapper who looks after the lead, and one helper to give the slag-, matte-, and lead-pots a clay wash, and to do other work.

171. **Work on the Dump.**—In former times, when little matte was made and the furnaces were small, there was in use the ordinary two-wheel slag-pot, Figs. 234 and 235. It is at present used as an auxiliary pot in ore-smelting and especially in the smelting of intermediary products in a refinery. The pot is wheeled out on the dump, and its contents are allowed to cool.[1] When these have solidified, the pot is tilted, and the cone of slag, with speiss and matte adhering to the bottom, rolled out. It is broken with a sledge, speiss and matte are sorted and piled, and slag thrown over the dump or reserved for further treatment.

At present, with large furnaces making much matte, the fore-hearth or settling reverberatory furnace collects any overflowing lead with the speiss and matte. The waste-slag pot, which catches the overflowing slag, is hauled by horse-, mule-, steam-, or electric-power to the edge of the dump and emptied;[2] in some instances the waste slag is granulated.

[1] Keller, *Tr. A. I. M. E.*, 1893, XXII, 576.

[2] Austin, *Tr. A. I. M. E.*, 1896, XXVI, 401.

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Until a few years ago slag-shells were saved and returned to the blast furnace. This practice was given up when it was proved by Wraith at Tooele, Utah, in 1915, that with sufficient settling facilities, the resmelting of slag did not recover enough values to pay for the cost. With the removal of slag-shells from the furnace, the percentage of coke required dropped from 12.92 to 10.76 per cent.; further, the smelting as a whole was improved, as with the slag-shells the zinc-content of the charge was increased. In Table 81 are given the values[1]

TABLE 81.—METAL VALUES OF SLAG AND SLAG-SHELLS AT TOOELE, UTAH

	Pb, per cent.	Cu, per cent.	Ag, ounces per ton	Au, ounces per ton
Original slag	1.1	0.28	0.31	Trace
Slag-shells	1.4	0.62	0.70	Trace
Original slag	0.7	0.20	0.38	0.005
Slag-shells	1.2	0.39	0.85	0.005

of the original slag, and of the slag-shells from the waste slag collected from two fore-hearths placed in series.

The number of men required on the dump varies with the size and number of furnaces, and the manner of disposal of slag. The tools required are steel bars, picks, sledges, round-pointed long-handle shovels, and iron two-wheel barrows.

172. Irregularities in the Blast-furnace.[2]—The disturbances that occur during the run of a blast furnace, having many different sources, are numerous. Some are caused by defective machinery and apparatus, others by refractory ores, faulty charges, or wrong manipulation. Some accidents due principally to the last-mentioned cause may be corrected in the following way.

The products of the furnace readily indicate the remedy which is to be applied to correct a faulty charge. It may happen that the top of the charge becomes hot when the furnace is doing good work otherwise. This does occur, if the charge contains battery residues which run high in PbO_2. This compound is readily dissociated by heat into PbO and O; and the O combining with C will generate heat. The remedy lies in heating the residues before they are charged, or in reducing the amount added so as not to exceed 10 per cent. of the charge. Kilbourn[3] tried to prevent a hot top by coating the coke with a fusible material rich in lead which does not give off O. A mixture of $PbSO_4$ and PbO with other lead-free substances which contained about 60 per cent. Pb worked satisfactorily.

When the charges do not descend as evenly as they should, one side sinking faster than the other, the jackets on the lower side being much hotter and the tuyères brighter than those on the upper side, the first change is made in the manner of feeding. The fuel is placed more on the hanging side, and ore and

[1] Pigott, *Eng. Min. J.*, 1916, CII, 626.
Anon., *Loc. cit.*, 1100.
[2] Lang, *Eng. Min. J.*, 1893, LVI, 662; 1894, LVII, 51.
[3] *Eng. Min. J.*, 1915, C, 313; U. S. Patent No. 1148782, Aug. 3, 1915.

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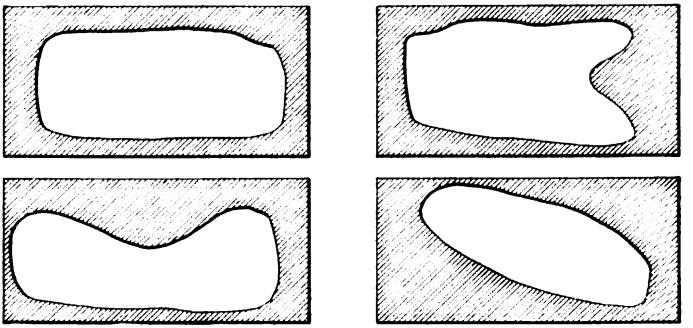
fluxes on the quickly descending one; then a full stream of water is turned into the hot jackets and that of the cool jackets reduced till there is just an overflow. By the combination of these remedies the smelting of the furnace on one side more than the other will be corrected, and the charges will right themselves again after a few hours. Shaking up the charge with a long, heavy (1½-in.) steel bar, introduced through the feed-door into the hanging side, will often hasten matters. At some works that have to treat ores rich in zinc, it is the practice at the beginning of each day-shift to drive in a steel bar at the four corners of the furnace as far as the top of the jackets, thus loosening any wall-accretions that are forming. By doing this, the number of times a furnace has to be barred down to remove accretions is greatly reduced, and the whole running improved.

When the charges descend irregularly, it often happens that the fire creeps up, and the charge becomes hot on the surface (overfire, fire-top, hot top).



FIGS. 349 to 352.—Wall accretions in blast furnace.

The furnace may then be fed down, which consists in adding only just enough fresh charge to keep the flame or heavy smoke from passing through. When the surface has thus been lowered for 2 or 3 ft., the furnace is filled up quickly again and the top thus cooled. Simple sprinkling of water on the top of the charge has only a temporary effect. This feeding down helps matters if the overfire does not come from a crust in the crucible.

The cause of the irregular descent of the charges lies generally in the formation of wall-accretions (§ 192), which begin on top of the jackets and grow thicker toward the feed-door. They assume different forms. Figs. 349–352, by Iles,[1] show some extremes. Fig. 349 represents a more or less regular shape, and the smelting power of the furnace is not necessarily reduced. With irregular hangings, like those in Figs. 350, 351, and 352, the descent of the charges will be greatly obstructed, and the amount of flue-dust much increased. The charge will be tight at the narrow parts of the furnace, and the blast entering the tuyères will be concentrated in a few places and cause "blow-holes." As soon as these wall-accretions are discovered, they have to be cut or barred down. In order to

[1] *Eng. Min. J.*, 1886, XLI, 93.

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reach the lowest part, the furnace is fed down and the blast lessened at the same time, till the charge has reached about the top of the jackets. While it is being lowered, no lead is removed from the well, in order that the crucible may be entirely full while the barring-out is going on. The blast is stopped, the blast-pipes are removed and the tuyère-openings closed, or the gates shut; all the slag in the furnace is tapped, and the flow of water into the jackets nearly shut off. When everything is ready, a charge of coke is given with some slag and flux, forming a bed for the accretions to fall on as they are chipped from the walls. The cutting-out is best begun just above the jackets. A square-pointed steel bar about 1¼ in. in diameter and long enough (about 18 ft.) to reach from the top of the jacket well into the opposite side on the feed-floor, is driven with a sledge into the crust above the jacket. If it does not yield, a rope, tied around the head of the bar, is thrown to the opposite side, where several men pull it and thus break off the crust. This is repeated until the crust has been removed in a number of layers, two sets of men working on opposite sides. While the crust is being barred off, fuel, slag, and fluxes are added that it may be smelted out, when the furnace is started up again. The reason that the barring down is begun from below is that otherwise the broken crusts and small slag-charges that have been added might so fill up the furnace that it would be impossible to reach the crust at the top of the jackets, and new accretions would form again quickly. Sometimes, however, if the crust is thick and hard, the barring down is begun from the top, continuing until the accumulated crust and slag-charge meet the clean side-wall, when these are smelted out, keeping the charges low. The furnace is now let down again, and the second half of the barring-down is begun at the top of the jackets and continued upward.

When the sides of the furnace have been cleared, the tuyères are cleaned out, the tuyère-pipes inserted, or the gates are opened, a weak blast, to be increased gradually, is turned on, and smelting resumed. Care must be taken about the water supply as the jackets grow hot quickly, and about the tapping of the slag, as the wall-accretions often melt very fast, and there is danger of the slag entering the tuyères. After starting the furnace, it is sometimes found that a small crust has formed over the lead while the blast was shut off. This is perforated with a long iron rod or with a steel bar, if necessary, and will soon disappear, if the furnace was in good working order below before the cutting out began.

A second method of barring down a furnace, said to work well, is to let down the charge to below the upper rim of the jackets, and give a bed of coke as previously described. In the meantime, the bricks between the jackets and the collar, on which the shaft rests, are removed for a distance of 1 or 1½ ft. The accretions are cut down, beginning from above, and raked out through this opening. When the shaft is clean, the opening is bricked up, light charges are given, the furnace is filled up with ore-charge, and the blast let on. The accretions go to the roasting furnaces.

A third method of barring down is dangerous, although it may be necessary at times and is even exclusively used by some metallurgists. The charge is let

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down to the top of the jackets, and cutting out begun from the top and continued until the crust collecting in the furnace has risen so high that the work cannot proceed any further. The two breast-jackets are now taken down and the contents of the furnace raked out, in order that the cutting may continue until the walls are clean. This may take as much as 18 hr. Two parties of three men each, working on opposite sides of the furnace, do the cutting, one man holding the bar and two sledging. As the work is hard and has to be done as quickly as possible, the regular hands have the constant assistance of furnace-men and helpers from other furnaces, who work half an hour at a time. While the accretions are being removed, a crust 6 in. thick or less will form on top of the lead. This is broken up, the breast-jackets are put back, and the lower front is closed. Fuel is fed from above to the top of the jackets and the furnace blown in anew, adding bars of lead to the first slag-charges to heat up the lead in the crucible.

Guyard[1] suggests that by using caustic lime in the charge instead of limestone these accretions might be made less troublesome, as lime has a decomposing action on the sulphides, of which they consist in great part. The use of burned lime as a basic flux has already been commented upon (§137). As to its desulphurizing action it should be remembered that while accretions next to the furnace walls consist mostly of sulphides, they are usually covered by a thin crust as hard as flint; this is followed by a softer substance that is often a powder, and this again covered by a crust so tough that it is sometimes extremely difficult for a steel bar, driven hard with a sledge, to produce any effect on it. It does not seem probable, therefore, that caustic lime would have any important effect on these composite crusts, which must differ from sulphides as much in their chemical properties as they do in their physical.

Some furnace-men[2] have found that charging a small amount of salt cake along the sides of the furnace frequently removed hangings.

It often happens, even when a furnace is otherwise doing well, that the tap-hole becomes hard and the tuyères dark. The fuel is then distributed more over the cold places, additional fuel being given for a short time if necessary. One or two bars of lead are sometimes placed on top of the charge over the tap or the tuyères, but this cannot be commended, although it is often effectual. The change in feeding the fuel will generally soften the tap. To soften a crust in front of a tuyère, the bag or the gate should be closed, or nearly closed, as the blast playing on chilled slag can have only a bad effect. If it is turned off, the heat generated at the neighboring tuyères will melt off the crust; then a little blast is allowed to pass through the tuyère, and gradually increased until the full blast can be turned on again.

When fine ore forms a larger part of the charge, more common formerly than is the case at present, a tuyère frequently becomes blocked. Placing an open tube through this[3] and thereby conducting the blast safely through the

[1] Emmons, "Geology and Mining Industry of Leadville," pp. 728, 747.

[2] *Eng. Min. J.*, 1905, LXXX, 932.

[3] Lang, *Eng. Min. J.*, 1890, XLIX, 174.

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crude ore may help matters. To remove the fine ore Lang[1] shuts off the blast from the tuyère and inserts a tube into the ore through the tuyère-pipe, when the inside pressure will blow out most of the fine ore, and this can be assisted by working the tube. This proceeding may also be necessary if some fine ore trickles through the coarser parts of the charge, appearing in a crude state at the mouth of the tuyère, and, as is sometimes the case, runs into the tuyère-pipe. If fine ore appears at the same time at several tuyères and too many would have to be tied up, the easiest remedy is to feed down the furnace and thus loosen the charge. If this does no good, a coarse ore-charge will have to be substituted for a short time, or if coarse ore is not to be had, a slag-charge.

The forming of hearth accretions is indicated on the feed-floor by the top becoming hot and the charges not sinking regularly, but in jerks, a foot at a time, and blazing up with every settling. On the furnace floor the lead in the well becomes dark, and does not play with the blast. By inserting a rod through the tap-hole, the position and often the thickness of the crust can be ascertained. Sometimes it only forms a ridge across the furnace, and communication is open in front and at the back; more commonly the crust begins at the back and grows toward the front, gradually closing all communication. The lead produced in the furnace cannot get into the crucible, and runs out with the slag. A case like this generally needs a change in the composition of the charge; the slag analysis will show the defect and give the remedy. As it takes from 6 to 8 hr. before a new slag makes its appearance, holes are driven with a steel bar through the crust that the lead may find its way into the crucible. If the crust has not yet hardened, it is repeatedly perforated by introducing an iron rod and working this up and down and to both sides. The crust may be only a temporary affair, and can then be worked into the slag, which removes it. Cases do occur where the crust will not yield quickly enough to being fused out and has to be removed by force. This has been ironically spoken of as "muscular smelting," but it is sometimes unavoidable. Before beginning, all the slag in the furnace is tapped, the basin of the lead-well covered with glowing coal, the blast stopped, the tuyère-pipes drawn out, the tuyères closed, and the flow of water into the jackets reduced. The breast of the furnace is now removed wholly or in part, any chilled slag in front is chipped off, and some of the loose material raked out into a wheelbarrow in front of the furnace and taken away. A heavy steel bar is passed through two opposite tuyères nearest the breast to hold up the charge. Balls of loam or clay are tamped behind the lower part of the front jacket to prevent the charge from rolling down. The balls, placed on the end of a board, are slid below the front of the jacket and rammed upward against the charge with a rod bent to a hook. When this is done, any lead that has accumulated in front is ladled out, and the crust thus laid bare. A hole is driven through it with a heavy steel bar. It is necessary to have a number of these ready, as the points soon become dull or bent. The hole is enlarged by driving the bar again close to it and breaking the crust toward it. When large enough to receive a 2-in. steel bar, this is warmed,

[1] *Loc. cit.*

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inserted into the hole, the crust pried up, and the broken pieces raked out. If the crust will not yield, the hole is enlarged and the furnace started up again. The lead previously bailed out is returned, and, if necessary, fresh hot lead added, the clay balls are removed, the hollow space in front is filled with fuel, the steel bars are withdrawn from the tuyères, the breast is put in, the tuyère-holes are opened, the pipes inserted, and the blast is let on, but very gently at first. When the first two or three pots of slag have been tapped the rod is repeatedly inserted to keep the hole open until the new slag comes down. The lead will soon show the effect of having communication between it and the slag partly restored; it begins to play with the blast and becomes hotter, thus assisting the work of the new charge.

A crust is sometimes caused by a leaking jacket. This is first indicated by the appearance of moisture at the tuyère or the bottom of the jacket. The leak, if small, can be temporarily stopped by mixing cornmeal with hot water, pressing it with the hand into small balls, and throwing these into the water-feeder of the jacket. Soon, however, the jacket will have to be removed. For this purpose the crust on the inside is first allowed to grow thick by cooling, which is done by turning in a full stream of water, and opening the discharge at the bottom. Two courses of brick are chiseled out above the jacket. When cool, the furnace is stopped, the cooling-water on the side of the injured jacket shut off, the water-trough removed, and the injured jacket unhinged, taken out, and a new one put in its place. The whole procedure need not take more than 20 min. Should the crust on the inside of the jacket prove too thin and break out, the opening is closed by the introduction of clay balls. The space where the new jacket is to be inserted must be absolutely clean, as any little pieces of brick or other hard matter will obstruct the placing of the new jacket and cause much delay. Of course the foregoing has no reference to a furnace with wrought-iron jackets extending its entire length.

The clogging-up of the lead-well is yet to be considered. In smelting sulphide ores rich in lead, sulphide of lead held in solution in the crucible often separates when the lead ascends the channel toward the basin. A bent iron rod may be inserted to clean it out. This presence of sulphides is generally caused by an incomplete decomposition of galena in the furnace on account of lack of heat in the smelting zone. If the charge is rich in copper, this causes coppery lead gradually to close up the channel.

173. Blowing-out.—If ore, flux, or fuel gives out, or if the furnace needs to be repaired, or if an accident happens that cannot be remedied in a short time, say in 18 or even 24 hr., the furnace has to be blown out. This is done by stopping the ore-charges and substituting slag-charges, until most of the ore has been smelted out. The charge is allowed to sink, and the blast is gradually lowered. Soon volumes of dark smoke mixed with white lead-fumes will appear. When the charge has receded somewhat, and before a flame makes its appearance, the damper in the flue leading to the dust-chamber is closed, and the fumes are conducted into the air by opening the damper on the top of the furnace or by lowering the sheet-iron stack, or by whatever contrivance may be in use

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for the purpose. If this were not done, an explosion might occur in the dust-chamber. To check the flame and to reduce the temperature, water is often sprinkled over the charge, although its effect on the lining of the furnace cannot but be deleterious. When the charge has sunk as far as the top of the jackets, the blast is stopped and the tuyère-pipes are removed. All the liquid slag is tapped, the tapping-jacket removed, and the breast of the furnace is knocked in.

Sometimes the furnace is blown down, allowing the charge to sink till only heavy fumes, but no flames, appear and the entire contents are then drawn. In this case there is no need of closing the damper to the dust-chamber; in fact, many furnaces have no damper at all.

The bulk of the slag remaining in the furnace is withdrawn with a hoe into iron wheelbarrows, emptied on the dump, and then chilled with water. As it is important that there should be little delay in drawing the charge, a number of wheelbarrows are placed one behind the other near the front of the furnace. As soon as the first is filled with red-hot charge, it is wheeled away and replaced by the second, the emptied wheelbarrow being put at the end of the line. When all the charge that can be easily reached with the hoe has been drawn, the front jackets are taken down, and the rest is removed. Meanwhile a thin crust will have formed on top of the lead in the crucible. This is easily broken, and the lead is then ladled into the molds, that have been moved from the lead-well to the front of the furnace.

Blowing down is more expensive than is generally believed. The items to be considered are: fuel remaining in the furnace, loss of half a shift in blowing out, dipping the bullion, barring out, sampling products, treatment of barrings, work of mason and extra fire-brick and fire-clay, warming the crucible, fuel for blowing-in, and loss of half a shift in blowing in. Beside, the general overhead expense of the works goes on in the same way, whether all the furnaces are working or only part of them. If a furnace has been blown down and is likely to be blown in again in a short time, it is advisable to keep the crucible warm, and a pole, 20 or more feet long, is inserted from the breast and kept burning slowly.

174. Furnace- and Assay-books.[1]—A daily record is kept of the work done by each furnace. One of the many suitable skeletons for this purpose is given in Table 82. The time when any change is made in the charge, or when anything out of the regular occurs, is noted in the first column of "Remarks." The second column of "Remarks" refers to the disposal of products, especially the shipment of lead bullion.

The assay-record, given in Table 83, furnishes information about the daily assays made of slag, matte and lead bullion. Under the head of "Remarks" are noted the names of any other furnace-products which may be assayed now and then. Tables 82 and 83 represent very simple forms for keeping records of the metallurgical work the furnaces are doing. In a plant there are kept similar records for each division of work. The keeping of accounts of a smeltery lies beyond the scope of this book.

[1] Argall, *Min. Sc. Press*, 1906, XCIII, 573, 722, 750.

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TABLE 82.—BLAST FURNACE RECORD

191 , November	Furnace No.	Charges No.	Composition of charge, lb.					Blower		Remarks	Input, lb.					Output							Remarks
			Ore mixture No.	Iron ore	Limestone	Slag	Coke	Rev. per min. No.	Pressure blast oz.		Ore mixture	Iron ore	Limestone	Slag	Coke	Slag		Matte		Lead			
																Pots	Lb.	Pots	Lb.	Pots	Lb.	Bars	

TABLE 83.—ASSAY RECORD

191 , November	Slag												Matte						Lead bullion			Remarks
	Per cent.											Oz. per ton	Per cent.				Oz. per ton		Per cent.	Oz. per ton		
	SiO_2	FeO	MnO	CaO	BaO	MgO	Al_2O_3	ZnO	S	Cu	Pb	Ag	Pb	Cu			Ag	Au	Pb	Ag	Au	

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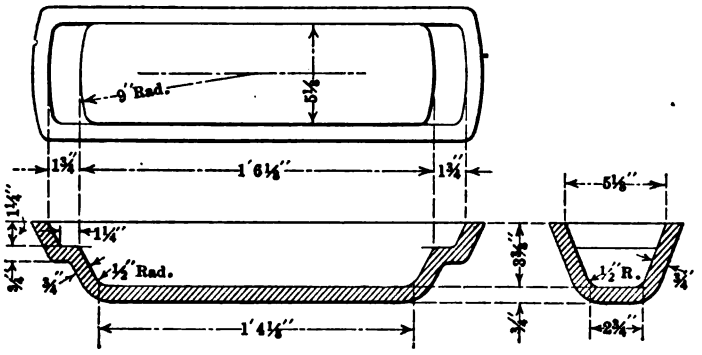
175. Products of the Blast Furnace.—The products of the blast furnace are soft lead and lead bullion, speiss, matte, wall-accretions, hearth accretions, furnace cleanings, and flue-dust; slag and gases have been treated in §117 and following, and §153 and following.

176. Soft Lead and Lead Bullion.—The term soft lead, as stated in §2, is used to designate the undesilverized lead of the Mississippi Valley; the name of lead bullion (or base bullion) is reserved for lead carrying precious metals. Both are cast into bars which are 3± ft. long, 4± in. wide, and 3±



FIGS. 353 to 356.—Mold for kettle dross.

in. deep weighing from 80 to 110 lb. The mold has the general form of the one shown in Figs. 353–356, which serves to mold drosses from the drossing kettle. It will be seen that the bar of dross is wider, shorter, and less deep than the bar of lead. A mold has in center of the bottom raised letters to denote the brand of lead or the name of the smeltery at which the lead has been produced.

TABLE 84.—ANALYSES OF LEAD BULLION

	Claus-thal	Frei-berg	Mech-ernich	Přibram	Leadville	Trail	Over-pelt	Hoboken-les-Anvers	Laur-ium	Port Pirie
Pb	98.80944	95.088	99.5913	97.3597	98.492379	98.86	98.78	94–96	98.50	98.80
Ag	0.1412	0.470	0.0215	0.4230	0.793417	0.412	0.14	0.3–0.5	0.190	.....
Au	.....	.....	.....	.....	0.000891	0.0013	0.14	.....	.....	0.044
Bi	0.0048	0.019	.....	0.0070	0.011791	.....	.....	.....	0.05	.....
Cu	0.1862	0.225	0.1332	0.1100	0.071450	0.05	0.19	0.2–2.0	0.50	0.34
Cd	Trace	.....	.....	.....	Trace	.....	.....	.....	.....	.....
As	0.0664	1.826	.....	0.2900	?	0.28	0.08	0.1–0.5	0.30	0.23
Sb	0.7203	0.958	0.2180	1.5240	0.347881	0.35	0.72	1–2	0.12	0.23
Sn	.....	1.354	.....	0.2500	0.000897	.....	.....	.....	.....	.....
Fe	0.0664	0.007	0.0300	0.0036	0.012600	0.01	0.04	.....	.....	.....
Zn	0.0028	0.002	0.0060	0.0012	0.000232	Trace	.....	.....	.....	0.05
Ni	0.0023	.....	Trace	0.0015	.....	.....	.....	.....	.....	.....
Co	0.00016	.....	.....	.....	.....	.....	.....	.....	.....	.....
S	.....	0.051	.....	0.0300	0.048934	.....	0.08	.....	.....	0.38
References	(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)

(1) *Zt. Berg. Hütten Sal. W. i. P.*, 1870, XVIII, 203. (2) *Wagner Jahresb.*, 1887, XXXIII, 401. (3) *Berg. Hüttenm. Z.*, 1886, XLV, 434. (4) *Oesterr. Jahrb.*, 1891, XXXIX, 52. (5) Emmons, *Geol. and Min. Ind.*, Leadville. (6) R. C. Rutherford, private communication, 1917. (7) *Ann. Min. Belg.*, 1901, VI, 270. (8) *Op. cit.*, p. 257. (9) Collins, *Eng. Min. J.*, 1905, LXXXIX, 363. (10) Poole, Sydney Univ Eng. Soc, Nov. 11, 1908.

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Compositions of soft lead have been given in Table 3; analyses of lead bullion are recorded in Table 84. The lead-content of the latter is usually 97 to 98 per cent.; it often falls to 95 per cent., and may reach even 90 per cent. with ores running high in Sb. The number of the other elements found in blast furnace lead is large. As regards their distribution in solid lead, they may be considered to fall into four classes, forming solid solutions, eutectic mixtures, chemical compounds, and having no affinity whatever for lead. The first ought to be uniformly distributed through a bar; the second[1] concentrated toward the center, which is supposed to solidify last; the position of the third will vary with the solubility; and the fourth will collect near or on the top. But some metals form both solid solutions and eutectic mixtures (Bi); others eutectics and chemical compounds (Te); again the eutectic points lie so near to pure lead that the latter retains a small amount of the foreign element (Cu); lastly the foreign elements may combine with one another independently of lead and thus add new complications. This irregular distribution of some foreign elements is brought out by the analyses of soft lead by Streng[2] and of lead bullion by Schertel[3] given in Table 85. Schertel kept the lead bullion for 24

TABLE 85.—DISTRIBUTION OF METALS AND SULPHUR IN LEAD

	Soft lead (Streng)				Lead bullion (Schertel)	
	Top	Bottom	Top	Bottom	Top	Bottom
Ag	.....	.....	.....	.....	0.423	0.403
Bi	.....	.....	.....	.....	0.132	0.042
Cu	3.621	1.242	0.508	0.140	1.324	0.034
As	.....	.....	.....	.....	2.164	1.980
Sb	0.274	0.158	0.090	0.057	0.700	0.749
Sn	.....	.....	.....	.....	0.941	(?)
Fe	0.008	0.008	0.012	0.008	0.103	0.009
Zn	0.003	Trace	0.002	Trace	0.016	0.003
Ni	0.148	0.082	0.012	Trace	0.029	.....
S	.....	.....	.....	.....	0.500	.....
Sp. gr.	.....	.....	.....	.....	10.321	10.824

hr. above the melting-point in an iron cylinder, 3 ft. 3⅜ in. high, before taking his samples. The results of Streng show that, together, Cu, Sb, Fe, Zn and Ni have a tendency to rise to the surface; those of Schertel indicate the same with the exception of Sb which he found to be concentrated at the bottom. Practical experience has taught that in a bar of lead, the impurities, with the exception of Ag and Au, are found to have collected nearer the top than the bottom, while the reverse is the case with Ag and Au. The results by the writer, given in Table 86 and Figs. 357–359, show that the lower part of a bar is richer than the upper, and that the center of the upper part represents the poorest

[1] Rosenlecher, *Berg. Hüttenm. Z.*, 1894, LIII, 333, 341.
[2] *Berg. Hüttenm. Z.*, 1859, XVIII, 14.
[3] *Wagner Jahresb.*, 1887, XXXIII, 401.

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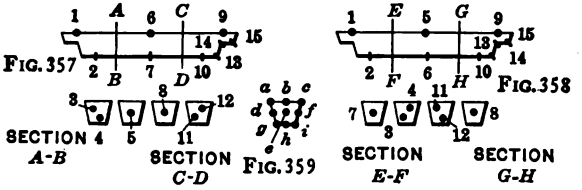
TABLE 86.—DISTRIBUTION OF AG IN LEAD BULLION

Fig. 357	(1.) 149.7	(2.) 150.5	(3.) 146.0	(4). 148.7	(5.) 145.0
	(6.) 137.0	(7.) 152.0	(8.) 149.0	(9.) 149.0	(10.) 151.0
	(11.) 148.0	(12.) 150.0	(13.) 150.5	(14.) 150.0	(15.) 152.0
Fig. 358	(1.) 127.0	(2.) 134.5	(3.) 128.5	(4.) 129.5	(5.) 125.0
	(6.) 133.5	(7.) 124.0	(8.) 132.0	(9.) 126.5	(10.) 134.5
	(11.) 129.0	(12.) 132.0	(13.) 133.0	(14.) 132.0	(15.) 134.0
Fig. 359	(*a*.) 432	(*b*.) 432		(*c*.) 439	
	(*d*.) 450	(*e*.) 434		(*f*.) 441	
	(*g*.) 385	(*h*.) 397		(*i*.) 386	

part. Similar results have been obtained by Piquet,[1] Raht,[2] Roberts[3] and others. Raht has shown that the part of a bar which solidifies last is the poorest in Ag. The bar represented in section in Fig. 360 was sprayed with water as soon as molded, that in Fig. 361, cast from the same bullion and at the



FIGS. 357 to 359.—Distribution of silver in lead bullion.

same time, was allowed to cool in the usual way. In Fig. 360 the shell, cooled by mold and spray-water, is the richest; in Fig. 361 the parts in contact with the heat-withdrawing iron are the richest.

An attempt has been made to explain[4] this concentration of the Ag in the lower part of the bar by the separation of the argentiferous lead, while cooling,

FIGS. 360 and 361.—Sections through bars of lead bullion.

into crystals low in Ag and liquid lead high in Ag. It is said that when a bar has been molded, the surface cools first, and crystals begin to form there; that the sides then slowly solidify, a mass of liquid lead and crystals remaining in the center. As the cooling is supposed to proceed from the top downward

[1] Roswag, "Désargentation du plomb," Dunod, Paris, 1884, p. 127.
[2] *Min. Ind.*, 1894, III, 414.
[3] *Tr. A. I. M. E.*, 1898, XXVIII, 413.
[4] Roswag, *loc. cit.*, p. 126.

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the crystals will continue to form on the cooler upper side and gradually force the richer liquid lead toward the bottom. This theory presupposes that the mold is a poorer conductor of heat than the air, which can not be the case. The reason that the surface of a bar is so much poorer than the other parts is not only because of the uneven cooling, but also on account of the impurities that rise to the surface. They run much lower in silver than the pure lead, as is shown in Table 87.[1]

TABLE 87.—WEIGHTS AND SILVER-ASSAYS OF LEAD BULLION AND ITS DROSS

Weight of lead, ounces	Weight of dross, ounces	Assay of lead, Ag, ounces per ton	Assay of dross, Ag, ounces per ton
26½	10¼	87.82	74.04
23¼	9½	92.05	74.47
27¾	7¼	86.59	63.55
65	15¾	84.21	66.47
31¼	11¼	94.84	77.30
30	10	87.79	74.40
28¼	7¼	93.30	75.56
23¼	10	95.96	76.44
24¼	11½	105.15	87.59
32	11½	89.62	77.10
54	24	82.91	67.56
51½	23	84.13	70.00
52½	14	106.24	78.92
45	17	87.24	68.80
47	12	98.69	79.25

If bullion is so rich in dross that the lead cannot eliquate freely, the natural distribution of the silver will be much disturbed. This will account for the fact that assays from the top of a bar sometimes run higher than those from the bottom. For instance, coppery bullion from the Ramshorn silver mine, Idaho, containing so much dross that if dropped on the floor it would break, gave to Rhodes[2] the results given in Table 86 under the head Fig. 359.

According to Kempf, Nenninger & Co.,[3] Au also seems to be concentrated with the Ag near the bottom of the bar as shown in Table 88.

Similar results have been published by Torrey and Eaton.[4]

TABLE 88.—RELATIVE DISTRIBUTION OF AG AND AU IN BAR OF LEAD BULLION

Sample from	Ag	Au	Ag	Au
Top	143.3	4.59	129.7	3.68
Bottom	148.4	4.76	134.3	3.82

[1] Private communication by W. Tatham.

[2] Private communication, July, 1891.

[3] *Eng. Min. J.*, 1882, XXXIV, 6.

[4] *Op. cit.*, 1886, XLII, 453.

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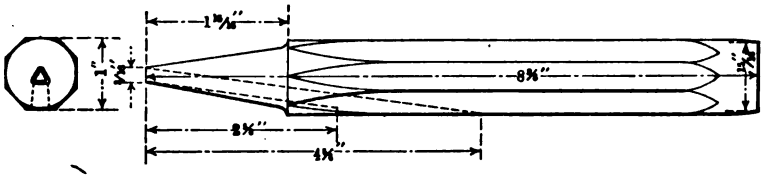
In the early eighties it was the common custom to bury the dross in the bars while they were being molded; at present this manner of disposing of dross has been given up. The lead bullion is melted at a low temperature in a kettle, poled with air or steam at a low temperature, the dross is skimmed, and the lead cast (§115 and 116). With the dross removed, the earlier large discrepancies in assay-values have disappeared. There remains, however, the unequal distribution of Ag, and with it of Au, which calls for definite modes of



FIGS. 362 and 363.—Sampling-punch for lead bullion.

sampling, if accurate valuations are to be made.[1] A method which unquestionably gives the best results[2] is to take dip-samples, cast in conical molds, so-called gum-drops, weighing from 15 to 40 gr. The lead bullion is melted in a kettle holding about 30 tons, drossed at 480° C., poled with air until the temperature has fallen to 360°, and skimmed on and off; the temperature is now raised to 380°, when dip-samples weighing about 0.5 assay-ton are taken, and assayed without trimming. In some cases a sample is taken for every bar that is cast. The samples are melted at a low temperature in a graphite crucible, the metal is stirred, and dip-samples are taken, as above, for the laboratory. Thus, a shipping lot of 42 tons contains 780 bars, and gives 780 "gum-drop" samples.

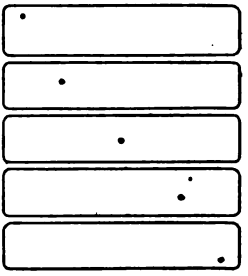


FIG. 364.—Punching of lead bullion.

With bars of lead bullion it is customary to take samples with a punch, such as is shown in Figs. 362 and 363. At a distance of $1\frac{15}{16}$ in. from the point it has a shoulder so that, when driven vertically into a bar, it will penetrate 2 in., or halfway, with the shoulder making an indentation of $\frac{1}{16}$ in. If driven halfway through the bar, say 2 in., it will represent a correct sample of that part of the bar. Sampling, however, from the ends of the bar gives too high a result, as was shown in Figs. 357 and 358. The poor parts of the bar, represented by No. 6 in Fig. 357 and No. 5 in Fig. 358, are excluded. The common method today is to take punch samples diagonally across a row of, say, five bars, as shown in Fig. 364, by driving the punch every time halfway through, turning over

[1] Keller, E. H., "Principles and Practice of Sampling Metallic Metallurgical Materials," *Bull.* 122, Bureau of Mines, Washington, 1916.

[2] Hulst, *Tr. A. I. M. E.*, 1914, XLIX, 539.
Weeks, *op. cit.*, 1915, LII, 714.
Easter, *op. cit.*, 729.

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the bars and repeating this on the other side in the opposite diagonal. In order to show where the holes are to be punched, it is advisable to use a templet of sheet iron fitting over the edges, and little spring punches which when tapped will mark the right spots on the bar. It will be noted that none of the spots fall either on the edge or in the median line. Two men usually work together, using 4-lb. sledges. One man will hold the punch vertically, while the other gives it the first blow, then both strike three or four times, whereupon a few taps on the sides loosen the punch and break the chip. In order to facilitate the breaking, the opening at the point is made sometimes three-sided (as in Fig. 362) instead of circular. Two men will sample a carload of lead bullion weighing from 17 to 20 tons in about 2 hr. One serious objection to this method of sampling is that on account of the flow of metal, the hardness of the dross, and the unevenness of the surface, the chips from the top of the bar are likely to be smaller than those from the bottom.

Recommendations by Kempf and Nenninger[1] and Austin[2] have not been followed.

Drilling instead of punching has been advocated, but has not found favor.

The chips obtained by sampling a carload or a lot are collected in a wooden box, melted, and cast into a sample-bar. A No. 10 graphite crucible is brought approximately to a white heat and taken from the furnace; the chips are dropped in and melt quickly; the metal is stirred and poured into an iron mold 2 in. thick which gives a bar, 9 by $3\frac{3}{4}$ by $\frac{1}{4}$ in., weighing 6 lb., or one 10 by 5 by $\frac{1}{2}$ in., weighing 10 lb., as shown in Fig. 365. The bar[3] is cut in two along the dotted line; one-half is reserved for the umpire assay; from the other half the four samples (Nos. 1 to 4), each weighing a little over $\frac{1}{2}$ assay-ton, are cut out to be assayed. It is essential that the iron mold should be cold and thick so that the bar, when cast, shall solidify quickly and the precious metals have no chance to become unevenly distributed.

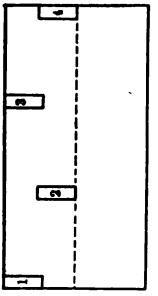

FIG. 365.—Sample-bar.

At the Delaware Lead Works the samples used to be melted at a very low temperature (some sawdust being added to make the rising dross as dry as possible), the lead was poured into a mold and the dross raked out on an iron plate and rubbed. Lead and dross were weighed and assayed and the result calculated as follows:

$$\frac{A.a+B.b}{A+B}=\text{Average assay.}$$

A = weight of dross; a = assay of dross; B = weight of lead; b = assay of lead.

A number of such weights and assays have been given in Table 87. Where a large quantity of lead bullion is desilverized, the method takes too much time. With bullion rich in dross, it is to be recommended.

[1] *Eng. Min. J.*, 1882, XXXIV, 6.

[2] *Op. cit.*, 133.

[3] Parmelee, *Met. Chem. Eng.*, 1916, XIV, 561.

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Sawing of bars as a means of sampling was suggested by Pattinson.[1] It was practised for a considerable time at Pueblo, Colo. It is used at the plant of the International Lead Refining Co.[2] with bars of dross cast from kettles in which lead bullion is melted and drossed previous to taking ladle-samples. The method is as follows: The bullion is unloaded from the car on to an inclined iron trough having rollers at the bottom over which it runs onto a table. Here a circular saw is so arranged with a slide and gauge as to make an incision halfway through the bar. The cuts are made respectively in the first bar at one-sixth of the bar-length, in the second at two-sixths and so on. The sample from a 20-ton lot weighing about 70 lb. is thoroughly mixed, quartered like an ore-sample, and then assayed in quantities of half an assay ton without any previous melting, as results obtained after melting are likely to be too low.

The further treatment of lead bullion, that is, desilverization, is taken up in Chapter X.

177. Speiss.—The speiss obtained in lead smelting is principally an arsenical speiss, as seen in Table 89. Antimonial speiss[3] is occasionally made in smelting softening skimmings (§262).

TABLE 89.—ANALYSES OF SPEISS

	Leadville, Colo.	Pueblo, Colo.	Denver, Colo.	Eureka, Nev.	Přibram, Bohemia	Monterey Mexico.	Overpelt, Belgium	Laurium, Greece
Ag	0.0085	Up to 0.014	0.044	0.029	0.037	0.049	0.017	0.109
Au	Trace	.....	0.0034	.....	.....	Trace	.....	.....
Cu	.3628	5.06	4.00	1.06	1.956	1.20	3.00	0.63
Pb	0.4935	0.69	2.00	2.18	1.752	3.50	1.90	0.50
Mo	01.2110	.....	.....	2.31	.....	.....	.....	.....
Fe	60.5780	59.42	56.80	57.02	61.330	71.60	55.00	63.80
Zn	Trace	.....	.....	0.07	.....	.....	2.50	1.75
Ni	0.0876	Trace	.....	.....	2.056	.....	5.10	.....
Co	.....	Trace	.....	.....	0.194	.....	5.10	.....
S	5.8191	2.80	.....	3.34	9.600	5.5	3.00	4.61
As	31.4725	31.17	32.50	32.95	18.750	15.60	23.60	20.75
Sb	Trace	Trace	.....	0.13	2.450	.....	5.90	.....
SiO_2	.....	Trace	.....	0.23	.....	.....	.....	0.80
CaO	.....	.....	.....	0.34	CaO 0.500	.....	.....	.....
Reference	(*a*)	(*b*)	(*c*)	(*d*)	(*e*)	(*f*)	(*g*)	(*h*)

(*a*) Emmons, "Geology and Mining Industry of Leadville, p. 720; (*b*) Dewey, Bulletin No. 42, "United States National Museum," p. 52; (*c*) Private communication of M. W. Iles; (*d*) Curtis, "Silver-Lead Deposits of Eureka, Nev.," Monograph VII, United States Geological Survey, 1884, p. 160; (*e*) Balling, *Berg. Hüttenm. Z.*, 1867, XXXVI, 419; (*f*) Hahn, *Tr. Inst. Min. Met.*, 1899–90, VII, 271; (*g*) *Ann. Min. Belg.*, 1901, VI, 270; (*h*) *Eng. Min. J.*, 1905, LXXIX, 364.

The predominant element in arsenical speiss is Fe; the other metals usually present are Ni, Co, Cu, and to a small extent Pb, Bi, Au, and Ag. Most speisses are produced in conjunction with matte; the S-contents of such speisses point to the solubility of matte in speiss.[4] Guyard[5] calls attention to the absence

[1] *J. Soc. Chem. Ind.*, 1892, XI, 321.

[2] Hulst, *Tr. A. I. M. E.*, 1914, XLIX, 539.

[3] Parravano-Cesaris, system As–Sb, *Internat. Zt. Metallogr.*, 1912, II, 70.

[4] Junker, freezing-point curve As–S, *Zt. anorg. Chem.*, 1909, LXII, 89.

[5] Emmons, *loc. cit.*, p. 721.

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of Co in Leadville speiss, which he found concentrated in the dross skimmed from the lead-well. He also found as much as 10 per cent. of grains of metallic Fe free from As in Leadville speiss, which is uncommon. It has been already stated that speiss always contains shots of Pb, and that when coarsely crystalline less is found than when it is fine-grained. As regards the presence of precious metals, the fact is to be noted that speiss retains considerable amounts of Au, whereas very little Au is found in matte; speiss-assays show from a trace to 0.5 oz. Au per ton. The presence of metallic Fe in speiss furnishes an explanation for the excess of Au over Ag usually found, as Fe has a strong affinity for Au.[1] Iles[2] gives as an average of 6 years' operation, that with ore assaying Ag 41.000 and Au 0.438 oz. per ton (ratio, 100 : 1.0683); the lead bullion assayed, Ag 266.00 and Au 3.49 oz. (ratio, 100 : 1.312); the matte, Ag 77.00 and Au 0.08 oz. (ratio, 100 : 0.104), Pb 10 and Cu 14.8 per cent.; and the speiss Ag 24.7 and Au 0.48 oz. (ratio, 100 : 1.94). An idea of the constitution of speiss is obtained by the study of the following binary alloys:

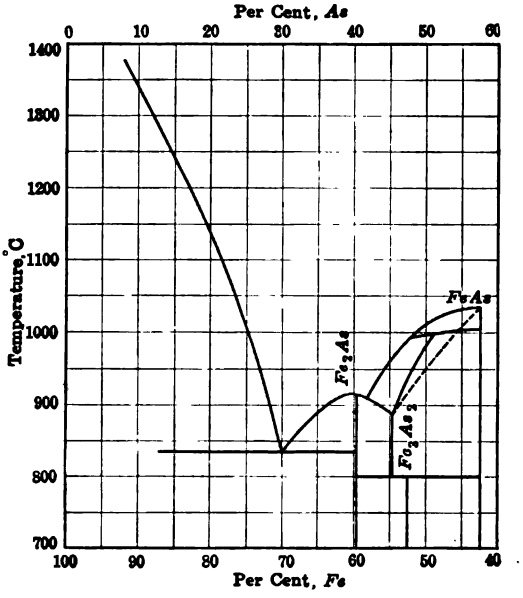

FIG. 366.—Alloy series Fe–As.

1. *Iron-arsenic.*—This system has been studied by Friedrich[3] within the limits of 91.6 and 44.0 per cent. Fe, and by Dieckman.[4] The curve of Friedrich, Fig. 366, shows the existence of the chemical compounds, Fe_2As, with 59.9 per cent. Fe, freezing at 919° C.; Fe_2As_2 with 53.2 per cent. Fe; and probably of FeAs with 42.8 per cent. Fe, freezing at 1030°; the compound Fe_2As_2 being the result of a reaction in the solidified alloy taking place at 800°; also the presence of an eutectic with As 30 per cent., freezing at 830° C., which consists of Fe_2As and Fe holding some As in solid solution. The alloys representing chemical compounds are extremely brittle; alloys with less than 60 per cent. Fe are not attracted by the magnet.

2. *Nickel-arsenic.*—According to Friedrich and Bennigson,[5] who investigated this system within the limits of 0 and 57.4 per cent. As, there exist, as

[1] See Hofman, "Copper," 1918, 215.
[2] "Lead Smelting," Wiley, New York, 1902, p. 129.
[3] *Metallurgie*, 1907, IV, 129.
[4] *Stahl u. Eisen.*, 1914, XXIV, 1694.
[5] *Metallurgie*, 1907, IV, 200.

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seen in Fig. 367, a solid solution of Ni and 5.5 per cent. As; the eutectic *a* with 27.8 per cent. As, freezing at 898° C.; the chemical compound *b*, Ni_5As_2 with 34.3 per cent. As, freezing at 998°; the eutectic, *c*, with 43.3 per cent. As, freezing at 804°; and the chemical compound, *d*, NiAs with 55.7 per cent. As, freezing at 968°. The compound Ni_5As_2 undergoes two changes, on cooling from 998°, forming solid solutions with 33.5 and 35.7 per cent. As; hence the eutectic *a* is composed of solid solution NiAs and compound Ni_5As_2 enriched in Ni; and eutectic *c* of compound Ni_5As_2 enriched in As and compound NiAs. In the solidified alloy there is probably formed at 800° the compound Ni_3As_2 with 45.5 per cent. As by a reaction between Ni_5As_2, enriched in As, and NiAs. The three chemical compounds are extremely brittle; only alloys with less than 29 per cent. As are attracted by the reagent.[1]

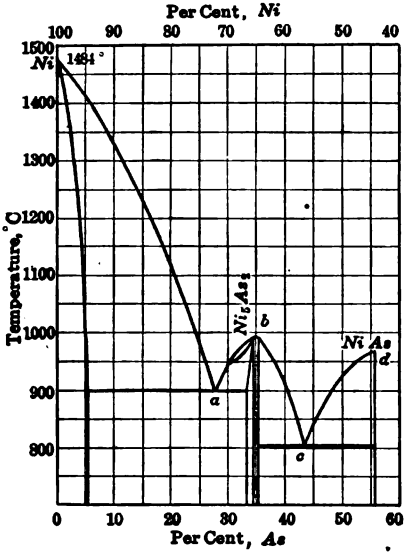

FIG. 367.—Alloy series Ni–As.

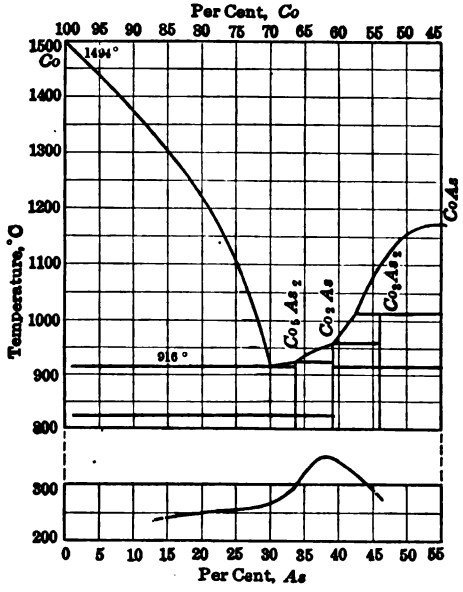

FIG. 368.—Alloy series Co–As.

3. *Cobalt-arsenic.*—The equilibrium diagram, Fig. 368, of Friedrich[2] shows that the system is complicated by several transformations. The metal Co, melting at 1494° C., forms a solid solution with up to 1 per cent. As; the curve shows an eutectic with 30 per cent. As, freezing at 916° and composed of solid solution Co–As and transformed Co_5As_2; also the following compounds: Co_5As_2 with 33.7 per cent. As, formed at 926° from Co_2As and liquid; Co_2As with 38.9 per cent. As, formed at 959° from Co_2As_3; Co_2As_3 with 45.9 per cent. As, formed at 1014° from CoAs; and CoAs freezing at 1179° C. Alloys with up to 38 per cent. As are not attracted by the magnet; Co_2As is non-magnetic.[3]

[1] See also Friedrich, *Metall. u. Erz.*, 1913, x, 659.

[2] *Metallurgie*, 1908, v, 150.

[3] See also Friedrich, *Metall. u. Erz.*, 1913, x, 659.

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4. *Copper-arsenic.*—The curves of Friedrich[1] and Bengaugh-Hill[2] are representatives for this system. In the revised curve of Friedrich, Fig. 369, Cu is seen to form with As a solid solution containing 4 per cent. As at 684° C.; then follow an eutectic with 78.5 per cent. Cu, consisting of solid solution Cu–As and Cu_3As; the compound Cu_3As, containing 71.8 per cent. Cu and freezing at 830°; and the hidden compound Cu_5As_2, with 67.9 per cent. Cu formed at 710°. No definite information exists regarding the horizontal lines at 604 and 307° C.

5. *Lead-arsenic.*—See §16.

6. *Other Metals-arsenic.*—Beside the principal metals of a speiss, the freezing-point curves of which have been given, there are likely to be present other metals in small quantities, such as Pt,[3] Au,[4] Ag,[5] Bi,[6] Sn,[7] Zn,[8] Mn[9] and others.

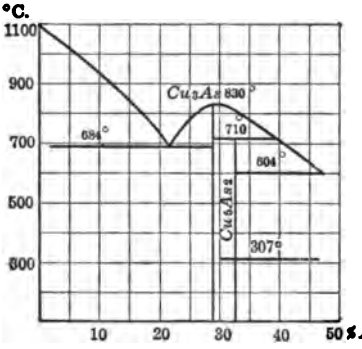

FIG. 369.—Alloy series Cu–As.

178. Treatment of Speiss.—The treatment of speiss in order to recover Pb, Cu, Ni, Ag, and Au has been and still is a difficult problem.

At Eureka, Nev., Davies[10] devised a simple method for the recovery of precious metals from the speiss given in Table 89. It consisted in tapping 800 lb. of speiss into a small cylindrical iron converter lined with fire-brick, Fig. 370, adding from 20 to 25 per cent. liquid lead and introducing from the bottom a blast of 17 oz. pressure through a ¼-in. pipe for 3 or 4 min. This stirred up the lead and speiss and burnt off some As. Most of the Ag and Au was taken up by the Pb; the liberated Fe corroded the lining somewhat, but ate through only very slowly. The converter was turned down and the content discharged into a cast-iron receiver having the form of a slag-pot, the bottom of which had a ½-in. hole for tapping the lead. The desilverized speiss solidified quickly, following which the still liquid lead was tapped. Best results were obtained when the lead assayed 40 oz. Ag to the ton. The desilverized speiss was then a waste product. The claim was made that from a speiss of the composition shown in Table 89, 85.5 per cent. of the Ag and 89.28 of the Au were extracted, with a loss of Pb varying

[1] *Metallurgie*, 1905, II, 484, 1908, V, 529.
[2] *J. Inst. Met.*, 1910, III, 34.
[3] Friedrich-Leroux, *Metallurgie*, 1908, V, 148.
[4] Friedrich, *op. cit.*, 1908, V, 603.
[5] Friedrich-Leroux, *op. cit.*, 1906, III, 192.
Heicke-Leroux, *Zt. anorg. Chem.*, 1915, XCII, 119.
[6] Friedrich-Leroux, *Metallurgie*, 1908, V, 148.
Heicke, *Internat. Zt. Metallogr.*, 1914, VI, 209.
[7] Parravano-Cesaris, *op. cit.*, 1912, II, 1.
[8] Friedrich-Leroux, *Metallurgie*, 1906, III, 477.
[9] Schoen, *op. cit.*, 1908, V, 737.
[10] Rickard, *Eng. Min. J.*, 1888, XLV, 474.

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from 5 to 8 per cent. At Freiberg, Saxony[1] and Oker, Harz Mountains[2] Pb-bearing speisses are treated in manners which meet local requirements.

The Freiberg speiss contains Fe 24.8, NiCo 19.6, Pb 7.1, Cu 10.9, Sb 27.0, As n. d., Ag 0.06, Au 0.0003, Pt 0.014 per cent. It is dead-roasted and then smelted with pyrite, galena, litharge low in Ag, and matte-slag in a lead blast furnace. Earlier work, smelting raw speiss in a reverberatory furnace, with the addition of an equal amount of flux consisting of 66 per cent. barite and 34 per cent. quartzite to form lead bullion, speiss, and matte, had shown that

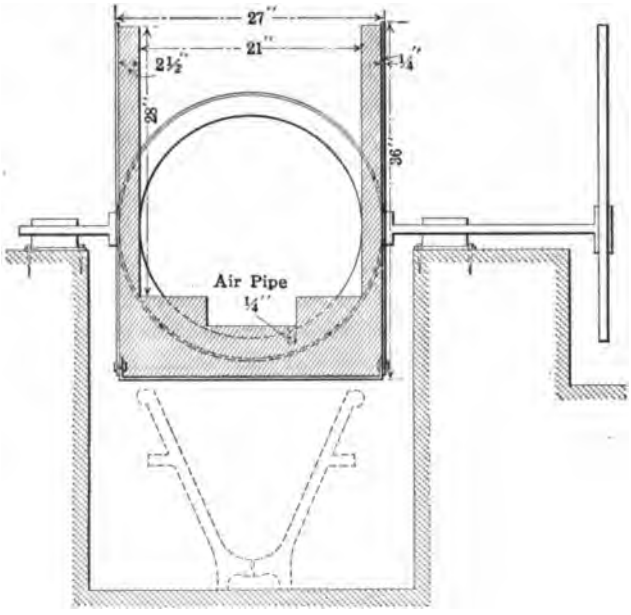

FIG. 370.—Davies converter for desilverizing speiss.

as long as the refined speiss assayed over 25 per cent. Fe, a matte containing Cu, Pb, Ag, and some Ni was obtained which separated readily from the speiss. The speiss, which amounts to 25 per cent. of the original charged, contains Fe 24.8, NiCo 19.6, Pb 7.1, Cu 10.9, Sb 27.0, As n. d., Ag 0.06, Au 0.0002, Pt 0.014 per cent.

The Oker speisses are represented by the following two analyses Fe 1.65, Zn 1.82, Ni 0.13, Co 0.24, Pb 35.20, Cu 51.73, Bi 1.63, Sb 3.34, As 2.75, S 1.38, AgAu 0.175 per cent.; and Fe 22.17, Zn 3.31, Ni 1.60, Co 1.11, Pb 16.68, Cu 25.85, Bi. n. d., Sb 13.50, As 11.04, S 4.13, AgAu 0.085 per cent.

The method of treatment is a roast-smelting in a reverberatory furnace. The products are copper to be granulated and slag. The copper granules

[1] Hübner, *Glück auf*, 1905, XLI, 6.
Hofman, *Min. Ind.*, 1905, XIV, 414.
[2] Huhn, *Glück auf*, 1905, XLI, 1165.
Hofman, *ibid.*

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have the following composition: Fe 0.028, NiCo 0.750, Pb 0.887, Cu 91.470, Bi 0.286, Sb 2.133, As 3.863, Ag 0.462, Au 0.0154; total 99.8944 per cent.; the slag goes back to the blast furnaces. There are in use two reverberatory furnaces, the hearths of which are built into iron pans, water-cooled at sides and bottom, with movable roof. One furnace has a hearth 10.33 by 5.74 ft. and a grate 5.58 by 1.64 ft.; the hearth of the other is 11.15 by 6.56 ft. and the grate 5.77 by 1.64 ft. Both furnaces have under-grate blast. The bottom of the hearth consists of a layer of marl tamped down firmly, followed by a course of brick laid dry. The working bottom, rammed into place, is a mixture of 20 parts shale and 80 parts marl ground to pass a 1-mm. hole. The metal bath of furnace No. 1 is 0.48 ft. deep, that of No. 2, 0.69 ft.; the copper is tapped from both sides into granulating basins. The mode of operation is as follows: Charge 3.5 tons poor speiss into the smaller, or 5 tons in the larger furnace; bring to a red heat and roast; melt when no more fumes are given off; skim the dross which has a metallic appearance, and turn on the blast (pressure, 38–40 mm. Hg); skim the slag, which takes up Fe, Sb, Pb, and Zn in the order given; lower the temperature, when slag ceases to form, until the bath becomes pasty, and turn off the blast to assist volatilization of As; fire up again when fumes of As cease to be evolved; continue cooling and firing up until no more As is volatilized. The speiss in the furnace has become enriched to 75 per cent. Cu. Charge 2.5–3 tons rich speiss; fire up; roast as much as possible and liquefy charge; skim the dross; turn on blast; skim slag; cool and reheat as before to drive off As. In order to obtain satisfactory granules, the Cu-content ought to range between 90 and 93 per cent. It takes 40 hr. and 1.43 tons coal to refine 6.5 tons charge in furnace No. 1; the 1.43 tons of hearth material, forming the working bottom, lasts 9 charges. With furnace No. 2, the 8-ton charge takes 48 hr. and 1.6 tons coal; the 1.6 tons hearth material of the working bottom lasts 6 charges.

Roasting of speiss with the object of expelling the As as As_2O_3, and using the Fe as a basic flux offers many difficulties on account of the formation of metallic arsenates ($Met_3As_2O_8$) which are not decomposed by heat. The reducing roasts advocated by Plattner[1] to convert arsenate into arsenide, do not appear to work satisfactorily,[2] and are rather tedious, lengthy, and thereby expensive. A very efficient method of expelling As is to crush the speiss, mix it with pyrite concentrate in the proportion of 10 : 1, and roast in a reverberatory furnace; the pyrite eliminates a large part of the As as As_2S_y, and the SO_3 formed in roasting decomposes MetAs, converting it into $MetSO_4$.

Heap-roasting in lots of 50 tons, which burn from 2 to 4 weeks, is a method which will be used only in exceptional cases. The erection of a heap requires special attention in order to insure a satisfactory draft; the product is uneven so that imperfectly roasted speiss has to be sorted out, crushed, and retreated in a reverberatory furnace.

[1] "Metallurgische Röstprocesse," p. 252.

[2] Pearce, *Tr. A. I. M. E.*, 1889–90, XVIII, 62, 457.
Howe-Campbell-Knight, *op. cit.*, 1907, XXXVIII, 162.

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Smelting partly-roasted speiss in a blast furnace with the addition of pyrite gives an excellent expulsion of As.

Blast roasting of speiss in a pot was first tried by Guillemain;[1] he crushed the speiss, mixed it with crushed limestone, silicious material (acid copper blast furnace slag, coarse sand, crushed fire-brick) and some lead matte for control of temperature and Cu-content. In blowing, he covered the grate with crushed limestone, charged glowing fuel, covered this with limestone, fed part of the charge to have the pot quarter-filled, and added the rest when oxidation had progressed satisfactorily. Thus, 12 tons speiss (Pb 9.9, Cu 6.65, Fe 48.5, NiCo 3.4, As 18.5, Sb 3.4, S 6.2 per cent., and Ag 61.2 oz. per ton), mixed with flux to reduce the As-content to 10–15 per cent., gave flue-dust (Cu 1.6, As 47.8, Sb 11.4, S 0.88 per cent., and Ag 1.6 oz. per ton) and, after fusion, a concentrated speiss with Pb 4.75, Cu 8.15, Fe 30.8, NiCo 12.5, As 20.6, Sb 0.90, S n. d. per cent., and Ag 72 oz. per ton.

An experimental charge of 500 lb. was blown for 5 hr., and its As-content reduced to from 3 to 5 per cent.

He found that leady speiss with 40 per cent. As (Pb 4.50, Cu 2.25, NiCo 0.70, As 4.00, Sb 0.30 per cent., and Ag 802.1 oz. per ton) had a higher melting-point and a behavior different from one with 20 per cent. As (Pb 2.75, Cu 1.00, NiCo 0.50, As 20.00, Sb 0.42 per cent., and Ag 300.40 oz. per ton). The speiss with 40 per cent. As emitted fumes of As_2O_3 and Sb_2O_3 until it had lost about 24 per cent. of its weight, was changed into a porous spongy cake, and required less flux than speiss with 20 per cent. As.

The Huntington-Heberlein process is in operation at Mapimi, Mexico. Speiss (Pb 12, Cu 9 per cent.) mixed with 2 to 3 times its weight of matte (Pb 10 to 16, Cu 10 to 14 per cent.) that the mixture may contain S 10 and As 7.5 per cent., is crushed to pass a ¼- to 5/16-in. screen, rough-roasted in a Godfrey furnace at the rate of from 24 to 30 tons in 24 hr., and blown in a 12-ton pot. The charge is made up of 50 per cent. Godfrey material, 40 per cent. silicious carbonate lead ore, and 10 per cent. flue-dust. The pot receives a bed of glowing coal as kindling; then 1-ton lots are fed at a time until 10 tons have been charged. The time of a blow ranges from 13 to 17 hr.; the blast roasted cake (S 6.4, As 5.0 per cent.) and the fines (10 per cent.) go to the blast furnace.

Converting of raw leady speiss, which usually contains about 5 per cent. S, in a small basic-lined converter with an addition of SiO_2 ought to work satisfactorily, as the amount of heat generated is more than sufficient for the purpose. Large-scale tests, however, have been unsatisfactory so far. The speiss appears to burn at the surface and attacks the lining, while on the bottom remains a heavy speiss rich in Pb and Cu which cools and forms a bottom-crust. Converting a mixture of matte and speiss can be made to work satisfactorily. In the absence of suitable matte, the speiss can be enriched in FeS and Cu_2S by smelting in the blast furnace with pyrite, FeS_2, and at the same time freed from

[1] *Metallurgie*, 1910, VII, 595.
Hahn, *Eng. Min. J.*, 1911, XCI, 858.

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a considerable portion of its As which passes off from the throat of the furnace as a yellow-brownish fume of As_xS_y.

In converting the speiss-matte, the Pb, a large part of the Fe, and some Cu and Ni go with the slag, while the concentrated speiss retains the remaining Ni, Cu, and Fe. The speiss may be granulated as it is poured from the converter, and treated by a wet process for the recovery of Ni, Cu, and precious metal.

179. Matte.—The matte produced in the lead blast furnace usually contains Fe 45–60, Pb 10–15, Cu 5–10, S 22–25 per cent. It is a mixture mainly of sulphides of Fe, Cu, and Pb, which frequently contains small quantities of metallic Fe, Cu, and Pb. Besides the main sulphides there are found sulphides of Zn, Ni, Co, Mn,[1] Ba, perhaps Na and K,[2] some arsenides, and slag. Many mattes contain some Fe_3O_4; this is more frequently the case with copper mattes,[3] formed in blast furnaces run with pyritic effect, than with lead mattes produced under strongly reducing conditions. The formation of Fe_3O_4 in smelting has been considered in §134 and 135. According to Mostowitsch[4] the magnetite in matte is present as a solid solution of FeO and Fe_3O_4, while Hostetter and Sosman[5] hold that there obtains a solid solution of Fe_3O_4 and Fe_2O_3.

TABLE 90.—ANALYSES OF MATTE

	Clausthal		Přibram	Overpelt	Pueblo			Denver		Leadville
	1867	1907								
Pb	41.50	13.72	11.16	8.00	10.72	8.50	16.00	8.00	14.8	18.0
Cu	0.36	7.68	1.59	4.85	0.61	20.40	43.50	3.00	9.5	10.4
As	.....	.....	0.55	3.00	0.56	Trace	0.14	.....	.....	.....
Sb	0.66	0.31	0.93	1.00	None	Trace	0.02	.....	.....	.....
Sn	.....	.....	Trace	.....	.....	0.16	0.175	Au 0.05(*a*)	Au 0.09(*a*)	Au 0.02(*a*)
Ag	0.12	.....	0.105	0.02	0.084	.....	.....	51.7(*a*)	95.7(*a*)	106.8(*a*)
Fe	34.05	44.73	41.31	57.00	52.27	41.20	12.30	37.5	31.6	47.6
Zn	.....	4.73	11.55	6.50	4.27	1.50	3.60	6.0	.....	.....
Mn	.....	.....	.....	.....	.....	1.40	1.20	.....	.....	.....
MnO	.....	.....	1.40	.....	.....	.....	.....	.....	.....	.....
Ni	.....	.....	Trace	0.70	.....	.....	.....	.....	.....	.....
Co	.....	.....	Trace	.....	.....	.....	.....	.....	.....	.....
SiO_2	.....	.....	3.06	.....	.....	0.20	2.20	7.8	4.2	1.2
CaO	.....	0.25	0.05	.....	0.41	.....	.....	.....	.....	.....
MgO	.....	.....	Trace	.....	0.47	.....	.....	.....	.....	.....
Al_2O_3	.....	1.01	Trace	.....	.....	.....	.....	.....	.....	.....
S	23.82	24.32	22.23	16.80	24.015	26.30	19.20	28.0	20.4	22.0
O	.....	.....	4.79	.....	.....	.....	.....	.....	.....	.....
Ref.	(1)	(2)	(3)	(4)	(5)	(5)	(5)	(6)	(6)	(7)

(*a*) = ounces per ton.
(1) Balling, "Metallhüttenkunde," p. 86. (2) Waldeck, "Streifzüge durch die Blei-und Silberhütten des Oberharzes," Knapp, Halle, 1907, p. 22. (3) *Oesterr. Jahrb.* 1891, XXXIX, 24. (4) *Ann. Min. Belg.*, 1901, VI, 270. (5) Private communication, W. H. Howard. (6) Same, M. W. Iles. (7) Same, R. D. Rhodes.

[1] Godshall, *Eng. Min. J.*, 1900, LXIX, 44.
[2] Canby, *op. cit.*, 1899, LXVIII, 3.
[3] Hofman, "Metallurgy of Copper," 1918, p. 214.
[4] Private communication, 1917.
[5] *J. Am. Chem. Soc.*, 1916, XXXVIII, 1188.

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Analyses of mattes are given in Table 90, which gives examples of normal and concentrated mattes. The high percentage of Pb in the matte produced at Clausthal in 1867 was caused by the area of the fusion zone, which was made large in order to obtain a low temperature and to diminish thereby the slagging of the brick furnace-walls; but the low temperature resulted in an imperfect decomposition of PbS. In the modern furnace, analysis of 1907, which has a contracted water-cooled melting zone, the lead-content is normal. The analyses show that, on the whole, the Pb-content rises with that of the Cu, whereas the percentage of Fe falls.

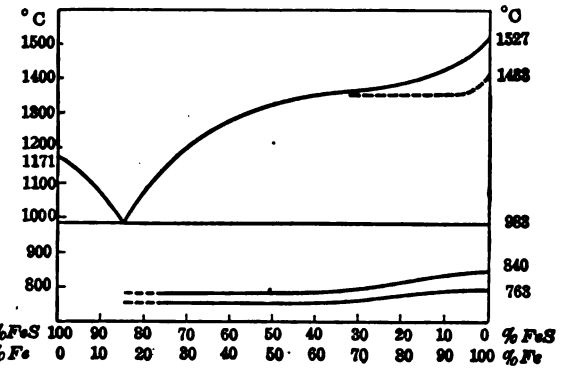

FIG. 371.—Alloy series FeS–Fe.

180. Constitution of Matte.[1]—The constitution of matte has been revealed to a certain extent by investigations into the binary systems of metal and sulphide, and of sulphides, which have been made in recent years. Ternary systems of sulphides, knowledge of which would supplement the results obtained by the work on the binary, have not been studied so far.

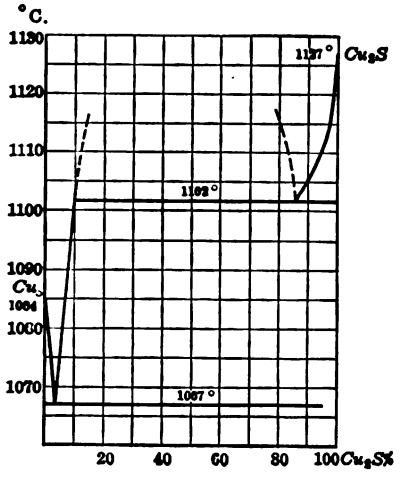

FIG. 372.—Alloy series Cu_2S–Cu.

1. *FeS–Fe.*—The freezing-point curve has been traced by Tammann-Treitschke[2] and Friedrich.[3] The curve of the latter is given in Fig. 371; it shows that the two components form an eutectic mixture with 85 per cent. FeS and 15 per cent. Fe, freezing at 983° C.; also that Fe can hold in solid solution less than 3 per cent. FeS, and FeS less than 1 per cent. Fe.

2. *Cu_2S–Cu.*—The curve of Heyn-Bauer[4] is given in Fig. 372. It shows that the freezing-point of Cu_2S, which lies at 1127° C., is lowered by additions of Cu until the temperature has fallen to 1102°; that a further separation of Cu from a mixture of the two components does not lower the temperature, until the composition of the original has been changed to 10 Cu_2S and 90 Cu, whereupon the curve, horizontal at 1102°, falls quickly to the eutectic point of 3.8 per cent. Cu_2S at 1067°, and rises similarly to the freezing-point of Cu at 1084°.

[1] Fulton-Goodner, *Tr. A. I. M. E.*, 1908, XXXIX, 584.

[2] *Zt. anorg. Chem.*, 1906, XLIX, 320; *Metallurgie*, 1907, IV, 54.

[3] *Op. cit.*, 1910, VII, 257.

[4] *Metallurgie*, 1906, III, 76.

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The more recent research of Urazoff[1] gives data differing largely from those of Bauer-Heyn; it states that the melting-point of Cu is lowered by additions of S, until an alloy with 4.36 per cent. Cu_2S has been formed; that it rises with further additions of S until a separation into two layers takes place, the upper freezing at 1121° C. and the lower at 1070°. He finds that liquid Cu_2S can dissolve 1.55 per cent. Cu and 1.12 per cent. S, and that Cu and S remain in solution upon solidification; but that on further cooling, the Cu_2S–Cu solution is transformed from β–Cu_2S to α–Cu_2S, and the Cu_2S–S solution assumes the more stable form of α–Cu_2S. The formation of moss copper is explained by the breaking-up of a solid solution having an excess of Cu, in the transformation of Cu_2S to another modification (see Ag_2S–Ag below).

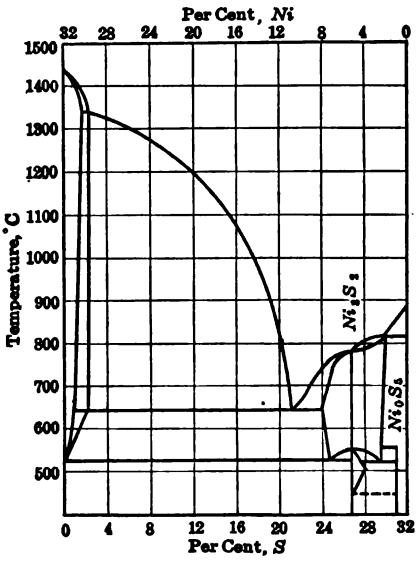

FIG. 373.—Alloy series Ni_3S_2-Ni.

3. *PbS–Pb.*—This curve has been discussed in §39.

4. *Ni_3S_2–Ni.*—The results of the study of Bornemann[2] of this series of alloys within the range of 0 and 31 per cent. S are given in Fig. 373. They show that the components form one chemical compound Ni_3S_2, freezing at 787° C., and probably a second Ni_6S_5; that there exists one eutectic of Ni–Ni_3S_2, freezing at 644° C. and probably a second, Ni_3S_2–Ni_6S_5. Between 0 and 26.7 per cent. S he found mixtures of Ni and Ni_3S_2; and between 26.7 and 31 per cent. S, mixtures of Ni_3S_2 and Ni_6S_5. His investigations as well as those of others[3] have shown that NiS heated in a neutral atmosphere gives off S until it has been changed to Ni_3S_2. In the older literature there is recorded the compound Ni_2S. Though Münster[4] and Schweder[5] recognized

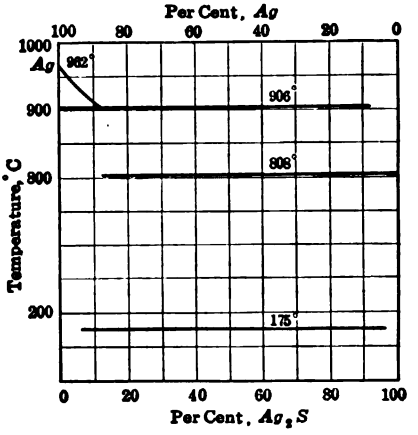

FIG. 374.—Alloy series Ag_2S–Ag.

[1] *Ann. Instit. Polytech. Pierre le Grand*, 1915, XXIII, 593, through *J. Inst. Met.*, 1915, XIV, 234.

[2] *Metallurgie*, 1908, V, 13; 1910, VII, 667.

[3] Hayward, *Tr. A. I. M. E.*, 1914, XLVIII, 141.

[4] *Berg. Hüttenm., Z.*, 1877, XXXVI, 195.

[5] *Op. cit.*, 1879, XXXVIII, 18.

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the existence of metallic Ni, and gave the mixture the formula Ni_2S, Bornemann has given the true explanation.

5. *Ag_2S–Ag.*—The curve of Friedrich-Leroux, Fig. 374,[1] extending from 0 to 97 per cent. Ag_2S, shows that the melting-point of Ag (962° C.) is lowered by admixtures of Ag_2S, until with from 11 to 15 per cent. Ag_2S it has fallen to 906°; with higher percentages of Ag_2S there are formed two layers extending to 90 per cent. Ag_2S. The horizontal line at 808° C. denotes the freezing of Ag_2S; the one at 175° a transformation of Ag_2S.

The more recent investigation of Urazoff[2] shows that liquid Ag_2S dissolves 0.65 per cent. Ag; that above this content there exist in the liquid state two layers, the upper freezing at 800°, the lower at 900° C; and that the lower upon solidification retains 14.7 per cent. Ag_2S. The research shows that Ag_2S undergoes a transformation at 576° which retains 0.5 per cent. Ag, but releases this, with lowering of temperature, at 175° and at 110° C. The release of Ag from Ag_2S furnishes one explanation for the formation of moss silver, but moss silver forms also from pure Ag_2S by dissociation.

The older research of Bissett[3] stated that Ag_2S melted at 815° C., that at 903° mixtures of Ag and Ag_2S containing from 17.25 to 49.24 per cent. Ag_2S formed two layers, and that there existed an eutectic with 99 per cent. Ag_2S, freezing at 804°.

6. *FeS–Cu_2S.*—The final answer to the constitution of this important system has not been given although it has been repeatedly examined.[4] Taking the freezing-points of FeS at 1155° C. and of Cu_2S at 1130°, which are the best figures at present, the two branches converge toward an eutectic point in the neighborhood of 870°, the actual place of which is not yet settled. Some investigators have found chemical compounds on the Cu_2S-branch, others deny their existence; again, some have failed to find any eutectic structure, others lay stress upon its existence.

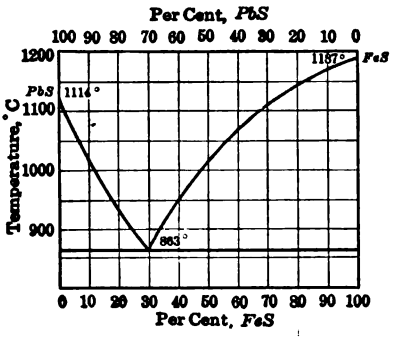

FIG. 375.—Alloy series FeS–PbS.

7. *FeS–PbS.*—This system has been studied by Weidmann[5] and Friedrich[6] with similar results although Friedrich chose as raw materials pyrrhotite (Fe_7S_8), and galena with 87.1 per cent. Pb instead of PbS with 86.6 per cent. The curve of Friedrich is reproduced in Fig. 375. It shows that the two sulphides form an eutectic mixture with 30 per cent. Fe_7S_8 and 70 per cent. PbS;

[1] *Metallurgie*, 1906, III, 361.
[2] *Loc. cit.*
[3] *Tr. Chem. Soc.*, 1914, CV, 1223.
[4] Hofman, "Metallurgy of Copper," 1918, p. 213.
[5] *Metallurgie*, 1906, III, 660.
[6] *Op. cit.*, 1907, IV, 480.

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freezing at 863° C. Weidmann places the eutectic point, with 25.8 per cent. FeS and 74.2 per cent. PbS, also at 863°. Both researches agree in drawing the eutectic line to the ordinates, thus proving that the sulphides are mutually insoluble in the solid state.

8. *FeS–ZnS.*—In the investigation Friedrich[1] used as raw materials pyrrhotite and blende. His curve, Fig. 376, covering a range of 0 to 60 per cent.

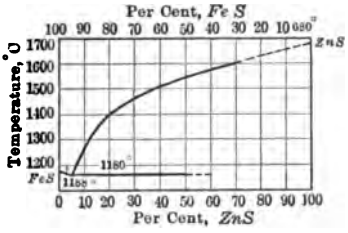

FIG. 376.—Alloy series FeS–ZnS.

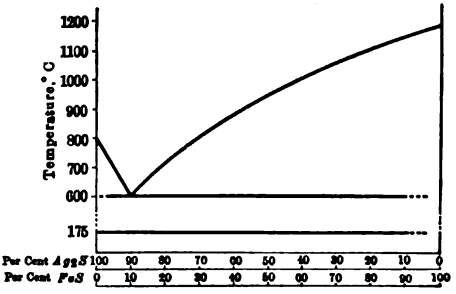

FIG. 377.—Alloy series FeS–Ag_2S.

ZnS, shows that the components form an eutectic with 95 per cent. FeS and 5 per cent. ZnS, freezing at 1188° C., and that ZnS is soluble in FeS to the extent of perhaps 3 per cent.

9. *FeS–Ag_2S.*—The diagram of Schoen,[2] given in Fig. 377, shows that the components form an eutectic with 11 per cent. FeS and 89 per cent. Ag_2S, freezing at 600° C., and that they are slightly soluble in one another so that within the limits of mattes usually produced, Ag_2S may be considered as

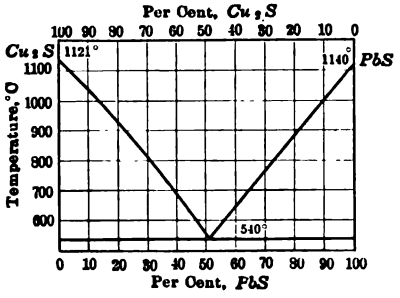

FIG. 378.—Alloy series Cu_2S–PbS.

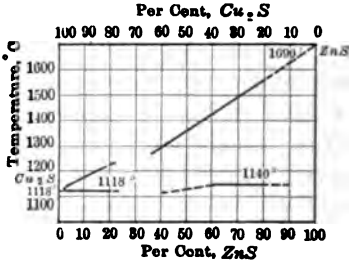

FIG. 379.—Alloy series Cu_2S–ZnS.

forming a solid solution with FeS. The horizontal line at 175° denotes the transformation of Ag_2S.

10. *Cu_2S–PbS.*—In the investigation of this system, Friedrich[3] used galena with 87.1 per cent. Pb, whereas PbS contains 86.6 per cent. Pb. His curve, Fig. 378, shows an eutectic with 51 per cent. Cu_2S and 49 per cent. PbS, freezing at

[1] *Op. cit.*, 1908, v, 114.
[2] *Op. cit.*, 1911, viii, 737.
[3] *Op. cit.*, 1907, iv, 671.

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540° C. The eutectic line extends from ordinate to ordinate, thus proving that the components are mutually insoluble.

11. *Cu_2S–ZnS.*—The zinc sulphide used by Friedrich[1] was blende with 66.39 per cent. Zn, whereas ZnS contains 67.08 per cent. Zn. His curve, Fig. 379, shows that the two components have no affinity whatever for one another and

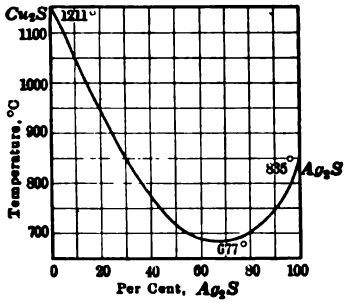

FIG. 380.—Alloy series Cu_2S–Ag_2S.

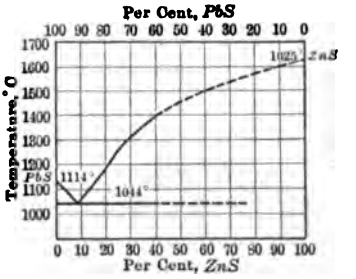

FIG. 381.—PbS–ZnS.

are mutually insoluble. It does not furnish an explanation of the fact that the presence of Cu in matte is favorable to the collection of ZnS. It has been shown above that ZnS is slightly soluble in FeS; it may be possible that the presence of Cu_2S increases the dissolving power of FeS.

12. *Cu_2S–Ag_2S.*—The curve of this system, Fig. 380, traced by Friedrich[2] shows that the components form solid solutions throughout; the minimum lies at 677° C. with an approximate composition of 30 per cent. Cu_2S and 70 per cent. Ag_2S. The curve gives the reason for the well-known fact that the presence of Cu_2S in a matte greatly favors the collection of Ag_2S.

13. *PbS–ZnS.*—The diagram of Friedrich,[3] based upon the use of blende and galena as noted above, is given in Fig. 381. It shows an eutectic with 92 per cent. PbS and 8 per cent. ZnS, freezing at 1044° C., and the insolubility of ZnS in PbS.

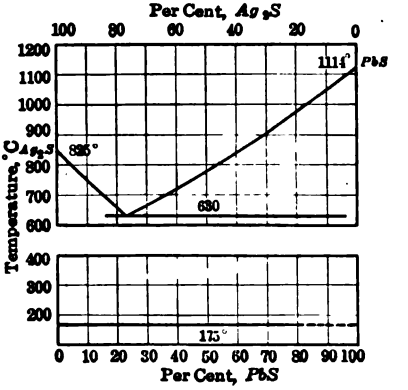

FIG. 382.—Alloy series PbS–Ag_2S.

14. *PbS–Ag_2S.*—Here Friedrich[4] used galena, as in previous work mentioned, as one component. The curve, Fig. 382, shows an eutectic with 23 per cent. PbS and 77 per cent. Ag_2S, freezing at 630°, and the solubilities of the com-

[1] *Op. cit.*, 1908, v, 114.
[2] *Op. cit.*, 1907, iv, 671.
[3] *Op. cit.*, 1908, v, 114.
[4] *Op. cit.*, 1907, iv, 479.

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ponents, Ag_2S forming with about 6 per cent. of PbS a solid solution, and PbS with about 3 per cent. of Ag_2S. The horizontal line at 175° indicates for the transformation of Ag_2S.

15. *PbS–Sb_2S_3.*—The results of the study of Wagemann[1] are given in Fig. 383. They show an eutectic composed of Sb_2S_3 and $2PbS.Sb_2S_3$ with 17 per

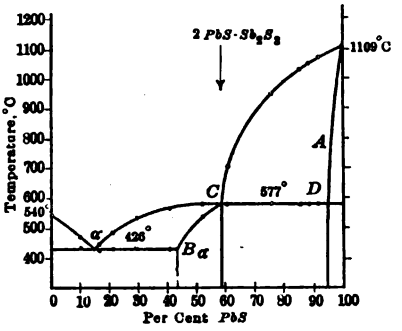

FIG. 383.—Alloy series PbS–Sb_2S_3.

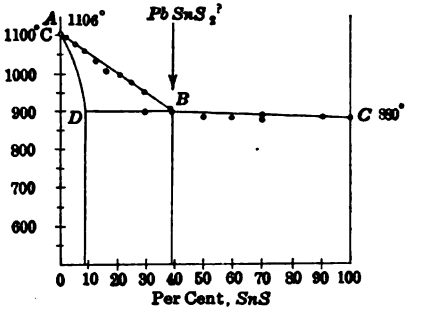

FIG. 384.—Alloy series PbS-SnS.

cent. PbS, freezing at 426° C.; the insolubility of PbS in Sb_2S_3; a solid solution of Sb_2S_3 and $2PbS.Sb_2S_3$ between 44 and 58.8 per cent. PbS, below 577°; the chemical compound $2PbS.Sb_2S_3$, freezing at 577°; and a solid solution of $2PbS.Sb_2S_3$ and PbS to the extent of 4.1 per cent. Pb at 577°.

The research of Jaeger[2] gives differing results. He found one hidden chemical compound $2PbS.Sb_2S_3$ at 609° (jamesonite), a second compound, $5PbS.4Sb_2S_3$ at 570° (plagionite), and an eutectic of plagionite and stibnite with 80 per cent. PbS, freezing at 495° C.

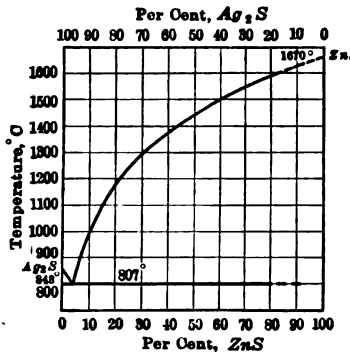

FIG. 385.—Alloy series ZnS–Ag_2S.

Pelabon[3] found an eutectic with 22 mol. PbS and 70 mol. Sb_2S_3 freezing at 482° C., one hidden chemical compound $PbS.Sb_2S_3$ (zinkenite) at 568° C., and a second compound $2PbS.Sb_2S_3$ (jamesonite) at 610°.

16. *PbS–SnS.*—The data of Heike[4] are given in Fig. 384. He finds that PbS, melting at 1106° C., forms a solid solution with as much as 8 per cent. SnS; and that SnS, melting at 880°, forms at its freezing temperature a solid solution with 61.27 per cent. PbS; he believes in the existence of the chemical compound $PbSnS_2$ with a melting-point of 880° C.

17. *ZnS–Ag_2S.*—The curve of Friedrich,[5] given in Fig. 385, shows an eutec-

[1] *Op. cit.*, 1912, IX, 518, 537.
[2] *VIII Internat. Congress Appl. Chem.*, 1912, II, 139.
[3] *Compt. rend.*, 1913, CLVI, 705.
[4] *Metallurgie*, 1912, IX, 313.
[5] *Op. cit.*, 1908, V, 114.

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tic with 3 per cent. ZnS and 98 per cent. Ag_2S, freezing at 800° C., and the insolubility of ZnS in Ag_2S; whether Ag_2S is soluble in ZnS has not been ascertained, as the eutectic line extends only to 80 per cent. ZnS.

18. *Arsenides.*—These have been treated in §117.

19. *Photomicrographs.*—Illustrations showing the constitutions of some mattes have been furnished by Fulton and Goodman.[1] Fig. 386 represents a concentrated blast furnace lead-copper-iron matte from Durango, Colo., with Cu 55.45, Pb 18.34, Fe 6.25, S 18.42 per cent., magnified 385 diameters. It shows an eutectic of PbS and Cu_2S, and a light field of the excess-component Cu_2S. The Cu_2S is probably a solid solution of Cu_2S and FeS; the dark specks in the light field are particles of metallic Cu. Fig. 387 represents a reverberatory

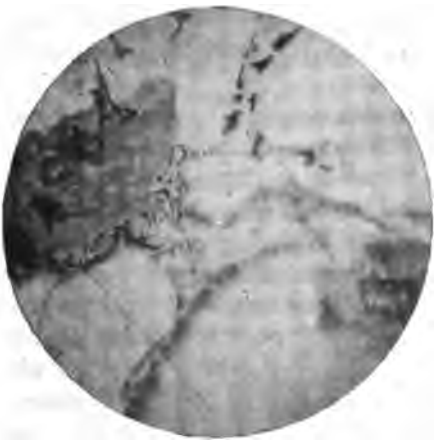

FIG. 386.—Lead-copper-iron matte.

FIG. 387.—Copper-lead-iron matte.

copper-lead-iron matte from Argo, Colo., with Cu 41.51, Pb 4.91, Fe 22.47, Zn 3.43, S 24.75 per cent., magnified 385 diameters. The center of the field shows the eutectic of PbS and Cu_2S; the light portions are excess component Cu_2S; the darker area at the bottom is FeS; particles of metallic Cu are visible in the light field of Cu_2S. Here again the Cu_2S may represent a solid solution of Cu_2S and FeS, and the FeS may contain some Fe.

20. *Matte as Collector of Precious Metal.*[2]—Lead matte is a good collector of precious metal; it carries considerable Ag, but little Au. It was formerly considered satisfactory, if the blast furnace matte assayed one-third as high in Ag as the lead bullion; at present it usually assays one-fifth as high unless it runs high in Cu. In concentrating matte by itself, about 50 per cent. of the Ag-content is collected in the lead bullion produced. The tendency of Au is to enter almost entirely the lead bullion. This is illustrated by a fact observed by F. R. Carpenter[3] that if pyritic matte containing little Cu is added raw to the

[1] *Tr. A. I. M. E.*, 1908, XXXIX, 540.

[2] See also, Hofman, "Metallurgy of Copper," 1918, p. 215.

[3] Private Communication, May, 1892.

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charge of a lead blast furnace, all the Au and from 60 to 70 per cent. of the Ag will be taken up by the lead bullion.

The laboratory experiments of Mostowitsch[1] have shown that the amount of Au and Ag extracted from matte by means of Pb is proportional to the amount of lead used; that with 80 lb. of lead to 100 lb. of matte, all the Au enters the lead bullion, and that with equal weights of lead and matte, only 72 per cent. of the Ag is recovered in the lead bullion. He explains the imperfect yield of Ag by the reaction $2Ag_2S+2Pb=Ag_2Pb+Ag_2S.PbS$; that is, Ag_2S is completely decomposed by Pb (see also §39).

From the binary systems studied, it is evident that Cu_2S is a powerful solvent for Ag_2S, and PbS and FeS have a considerable tendency in the same direction. As regards Au, a good solvent is found in Cu_2S; the presence of Ag_2S appears to act favorably upon carrying Au into matte; the other sulphides have not much effect. However, metals, such as Pb, Cu, and Fe have a strong tendency to combine with Au.

21. *Nickel Matte.*—The blast furnace matte usually contains little Ni and Co. A case of special interest is that of the non-argentiferous lead ores of Southeast Missouri which carry small amounts of Ni and Co which are concentrated in the matte when smelted in the blast furnace. At Mine la Motte[2] the first matte carries from 3 to 3.5 per cent. NiCo, from 0.5 to 1 per cent. Cu, and from 20 to 25 per cent. Pb. When this is roasted in a reverberatory furnace, there must remain, when it comes to be used as iron flux, from 5 to 6 per cent. S in the matte, if the loss of Ni and Co by slagging is to be avoided. In the same way the second matte (5–6 NiCo; 20–30 Pb; 1–2 Cu), when roasted, requires from 7 to 9 per cent. S to reduce the loss by slagging; it is then smelted with silicious matter to a third matte (12–17 NiCo; 35–40 Pb; 3–5 Cu), the slags nevertheless assaying 1.25 per cent. NiCo and from 2 to 2.5 per cent. Pb. Attempts at further concentration in a reverberatory furnace proved unsuccessful, the slags carrying 5 per cent. NiCo and 5 per cent. Pb. This illustrates the difficulty of concentrating NiCo in a matte when Pb is present, which apparently drives them into the slag. The Ni can then be recovered only in part and the Co not at all. There remains the introduction of As to form a speiss. Neill used this method successfully. The experimental furnace he built was 30 in. wide at the tuyères, and 48 in. at the charging-door; the height from tuyères to charging-door was 6 ft. 6 in.; the crucible, 18 in. deep, was partly internal, partly external, and had a siphon-tap. He obtained a speiss with 22.53 per cent. NiCo (one-third of which was Co), 6.4 per cent. Pb, and 4.25 per cent. Cu. The matte that formed contained 3.25 per cent. NiCo, 8 per cent. Pb, and 7 per cent. Cu, to be retreated; the slag from 0.16 to 0.32 per cent. NiCo, and from 0.6 to 0.8 per cent. Pb.

181. Treatment of Matte, General.—The object of treating matte is to extract the Ag and Au by means of Pb, and to concentrate the Cu in a small amount of Cu-rich matte which is either sold to copper works or brought

[1] *Met. Chem. Eng.*, 1916, XIV, 703.

[2] Neill, *Tr. A. I. M. E.*, 1884–85, XIII, 634.

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forward to metallic copper to be desilverized by vitriolization or electrolytic refining. Electrolysis of matte[1] has been tried, but has not been successful.

All concentration-smelting of matte from a lead blast furnace has to be preceded by an oxidizing roast in order to eliminate some S and to oxidize some Fe that it may combine with SiO_2 and enter the slag. Pyritic smelting,[2] as practised with copper matte, cannot be applied to lead matte on account of excessive losses in Pb, and with this of Ag, which are unavoidable. The roasted or blast roasted matte is smelted in the lead blast furnace until a matte with about 14 per cent. Cu is obtained. The further treatment may be carried on in the blast furnace, the reverberatory furnace, or the converter.

182. Roasting and Blast roasting of Matte.—Roasting may be carried on in heaps, stalls, kilns, and reverberatory furnaces. The choice of method will depend mainly upon the value of the matte, and the cost of operating; a rich matte must be roasted quickly so as not to lock up too much money for several months.

In roasted matte,[3] the external part of each piece or pellet contains more Ag and Pb than the interior. Any dust that is made in handling roasted matte, coming from the surface, will cause considerable loss; again, if exposed to rain, Ag_2SO_4, formed in roasting, is likely to be lost by leaching.

Roasting in heaps[4] and stalls has become obsolete in this country; it is still in operation in some European plants, but is bound to be replaced there by blast roasting as has been the case here.

Roasting in kilns, carried on in works of the Harz Mountains, Prussia, and of Freiberg, Saxony, which are forced to take care of the sulphurous gases by converting them into sulphuric acid, has not been practised here, as far as the writer is aware. Details of these three roasting methods are contained in the preceding edition of this book.

Roasting matte in the reverberatory furnace, which was common practice until within a few years, has been replaced almost wholly by blast roasting with silicious material as flux, on account of the superiority of this process as regards cheapness, small loss in metal, and the advantage accruing from smelting blast roasted lumps of oxidized matte and flux rather than the pulverulent product of the reverberatory furnace. Hand reverberatory furnaces used to be in greater favor than mechanical roasters, as the former gave a better elimination of S, and the fusibility of matte caused much caking on the hearths of the latter. Some hand reverberatories are still in operation in spite of their low tonnages and high costs of operation. Little need be added to the facts recorded in Table 42 as regards the hand reverberatory furnace.

Descriptions of mechanical reverberatory roasting furnaces and data of their work are given in the preceding edition of this book and in the writer's "Metallurgy of Copper."[5]

[1] Hofman, "Metallurgy of Copper," 1918, p. 482.

[2] Hofman, *op. cit.*, p. 189.

[3] Plattner, "Die Metallurgischen Röstprocesse," Freiberg, 1856, p. 205.

[4] Canby, *Eng. Min. J.*, 1908, LXXXV, 719; formation of sulphide minerals.

[5] McGraw-Hill Book Co., Inc., New York, 1918.

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In recent years the Wedge roasting kiln has overcome, by careful regulation of temperature, the fusion and thereby the adhesion of matte particles to the hearth. This furnace and its work have been considered in §92.

Blast roasting matte. The average composition of the first blast furnace matte has been given in §178 and Table 90. It shows that the matte must be readily fusible, and that its S-content is too high for a satisfactory blast roast. The reduction in S is accomplished either by rough-roasting the matte and adding the fluxes necessary for smelting, or adding to the raw matte a diluent (silicious material) to the extent of about 20 per cent., which will counteract premature fusion and reduce the S-content to the amount required for blast roasting. As matte and flux have to be intimately mixed, it is necessary to reduce both to a suitable size, say $\frac{5}{16}$ in. and smaller.

Matte is obtained from the blast furnace either in the form of a solid cone or cake, which has to be crushed, or in the liquid state, when it has to be granulated. Cakes of matte rich in Fe disintegrate readily when exposed for a few days to air, especially when this is moist; they are crushed in breakers and rolls, usually set to $\frac{5}{16}$ in. Granulation has been discussed in §119. There is no question that the angular grains of crushed matte roast more readily than the rounded pellets obtained from granulation; the difference, however, diminishes with the size of particles, so that with pellets of $\frac{3}{16}$ in. diameter it ceases to be noticeable. But in granulating, a considerable portion of the pellets is larger than $\frac{3}{16}$ in., and these are imperfectly roasted. Many smelteries prefer to incur the heavier expense of crushing in order to obtain a product more suitable for roasting. In the charges for blast roasting ore, given in Tables 46 and 47, it is shown that the matte is usually incorporated in the ore mixture. The principles and practice of blast roasting matte alone mixed with silicious material do not differ from those which govern the treatment of ore, and these have been given in §84–101.

A new departure has been made at Herculaneum, Mo.[1] The granulated matte (§117), with particles not exceeding ½ in. in size, is mixed with an equal volume of limestone (dolomite), crushed to ½ in., or concentrator tailing (chats), thoroughly wetted and blast roasted in a Dwight-Lloyd machine run at a speed to treat in 24 hr. about 75 tons charge. There results a product, well sintered, with 4 to 5 per cent. S. An analysis gave, Pb 10.39, Cu 1.64, Insol. 3.3, SiO_2 2.35, FeO 44.24, CaO 16.05, MgO 9.28, S 4.70, Zn 3.59 per cent.

183. Concentration Smelting of Roasted and Blast roasted Matte in the Blast Furnace.—In adding roasted matte with, say, 5 per cent. Cu to the charge of the ore blast furnace for the purpose of utilizing the FeO as basic flux and producing a matte with about 14 per cent. Cu, there have occurred troubles from fines. This has been especially the case, if roasted 14-per cent. copper matte is to be enriched to 40 to 50 per cent. Cu by smelting with silicious flux and limestone. Partial analyses of concentrated matte are given in Table 91. A simple and conclusive explanation of this trouble has been fur-

[1] Private notes, July, 1916; *Eng. Min. J.*, 1916, CI, 943.

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nished by Palmer[1] who calls attention to the fact that in smelting a mixture of coarse and fine materials, the fine sifts through the coarse and arrives first at the fusion zone, causes irregular working of the furnace, and gives rise to crucible troubles. The present practice, as stated above, is either to rough-roast the matte and blast roast it with silicious fines, or to mix it raw with silicious material and blast roast direct. The choice between the two methods will be governed by the character of the silicious material and the rest of the charge. Thus often fine oxide ores may be advantageously added to the charge in sufficient quantity to reduce the S-content to the amount required for successful blast roasting.

The blast furnace charge must not be made up entirely of coarse material, as it becomes too loose or open, resulting in too rapid smelting and consequent imperfect reduction. Blast roasting, however, furnishes, Tables 46 and 47, from 10 to 15 per cent. fine material which has approximately the same slag-forming constituents as the 95 to 85 per cent. coarse. The fines added to the coarse charge give it a correct denseness, and being about of the same composition as the coarse part do little harm should they separate. In any case, their amount is so small that they are well held up by the coarse, and do not segregate to any noticeable degree.

TABLE 91.—ANALYSES OF CONCENTRATED MATTE

Cu	Fe	Pb	Zn	S	Ag[2]	Au[2]
17.9	28.7	18.8	.....	17.4	.....	.....
18.4	28.0	17.2	.....	20.8	.....	.....
18.6	25.6	17.8	.....	16.5	.....	.....
37.6	22.6	5.3	7.8	24.0	7.921	0.0534
38.3	9.3	38.8	.....	13.3	.....	.....
42.0	23.3	4.7	4.6	22.0	13.037	0.1438
43.7	20.8	4.7	4.0	22.0	15.162	0.1424
44.3	19.8	6.5	5.1	20.5	10.324	0.0992
44.8	16.1	10.5	4.9	20.0	7.593	0.0847
48.3	5.9	23.5	.....	15.1	.....	.....

The reasons for limiting matte-concentration in a lead blast furnace with siphon tap to a matte with about 14 per cent. Cu have been given in §147. The mode of operating is the same as when a matte of lower grade is being produced in the regular ore-smelting. There is, however, this difference in the fusion zone, that a matte with up to about 30 per cent. Cu is very hot, fluid, and corrosive, so that more water is required in the jackets than in ore-smelting; it makes the slag run freely, and this eats into the brickwork below the jackets and into the clay-plug stopping the tap-hole. These may have to be freely sprayed with water. On account of the rapidity with which a matte-charge passes through the furnace, the volume of blast used is often only one-third of that customary

[1] *Tr. A. I. M. E.*, 1914, XLIX, 507.

[2] Ounces per ton.

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with an ore-charge. The Cu-content of the matte is limited to 50 per cent., as with a higher concentration metallic Cu begins to separate.

A plant treating lead ores usually does not have sufficient copper-bearing material to warrant a separate matting blast furnace; a lead blast furnace therefore has to serve for concentration smelting to produce matte with 40 to 50 per cent. Cu.

The simplest way of operating is to change from ore-charge to matte-charge, to work in the usual way, reducing the volume of air as the furnace otherwise becomes hot at the top, and to collect lead bullion from the well until this freezes or becomes choked with coppery lead. Then lead bullion and matte are collected in the fore-hearth, with slag overflowing in the usual way. Lead bullion and matte are tapped into an iron tapping basin; into the liquid contents is placed an inverted T-shaped iron bar with hook or eye so as to permit raising later of the solidified cake of matte by means of a tackle, and molding the lead. When the matte on hand has been smelted, the furnace is put again on ore-charge. The changes in lead bullion and matte are ascertained by assays. As soon as these show normal values, the lead-well is opened with an oil flame, and loosened dross is raked out. When the bottom of the channel has been reached, a hot bar is driven through the remaining plug to furnish an opening for the lead. This will rise in the channel and carry with it the coppery lead still remaining in the crucible. It takes about a week for the lead-well to close up, and a fortnight to remove completely the coppery lead from the crucible. At some works the tapping jacket has two tap-holes one above the other. The lower may serve for lead and matte and the upper for slag, or the lower for lead alone and the upper for matte and slag.

Furnaces are also provided with a lead-tap on the side opposite the well when matte and slag are tapped at the front in the usual way. The lead-tap is then placed from 3 to 8 in. below the top of the crucible.

In order to reduce the amount of intermediary grade of lead, the lead in the crucible is dipped through the well until the lower end of the channel is laid bare. This is then closed with a clay plug, and ore run until the crucible is about filled again, when matte-charge is put on. The lead reduced from the matte is removed at intervals through the lead-tap. When the matte on hand has been smelted, ore-charge is put on the furnace, with extra lead if necessary. As soon as lead begins to come quickly, the clay plug in the siphon-tap is removed and the furnace run in the usual way. During the matte-run some copper enters the lead, but being kept sufficiently hot, it does not solidify, and smelting goes on satisfactorily on a bath of mushy lead. In the course of 1 or 2 days after the ore-charge has been put on, the coppery lead will have been replaced by fresh lead from the ore-charge.

At a few plants the crucible of a cold lead blast furnace is filled with well beaten brasque and thus changed into a matting furnace. In such a case, it is advisable to have the new bottom of the furnace inclined. Thus the distance between center of tuyère and slag-tap is 10 in. and the tap-hole for matte and lead is placed 8 in. lower at the opposite end of the furnace. This furnishes a

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excellent separation of matte and slag. In this connection it is important to collect every scrap of lead contained in the brasque when this is being removed, as the lead is likely to run very high in precious metal. For example, in making matte with 30 per cent. Cu, 600 oz. Ag and 2 oz. Au per ton, the lead soaked into brasque assayed 3447 oz. Ag and 222 oz. Au per ton.

The slags made are similar to ore-furnace slags, with this difference, that they usually run lower in CaO and are not quite so silicious. Table 92 gives a few examples; they will show less than 0.5 oz. Ag per ton and contain from 1 to 1.5 per cent. Pb. In order to reduce the lead satisfactorily, it is often necessary to use a high percentage of coke, and scrap iron forms a valuable addition to the charge.

TABLE 92.—MATTE-CONCENTRATION SLAGS

SiO_2	FeO	CaO
32	36	14
32	40	16
33	58	9
30	40	10

A furnace, 36 by 108 in. at tuyères and 12 ft. high, will put through in 24 hr. with from 10 to 12 per cent. coke about 110 tons matte-charge.

184. Concentration Smelting of Roasted Matte in the Reverberatory Furnace.—The reverberatory furnace is used only with matte containing from 40 to 50 per cent. Cu, *i.e.*, at the point where the blast furnace has to stop; but is not found now as often as was the case formerly, as it has been replaced by the basic converter.

The common practice was to crush and roast 40- to 50-per cent. copper matte and smelt it with raw matte and some silicious flux, producing matte with 70 per cent. Cu, and a slag which carried the Fe and Pb. It was often necessary to flap the matte to insure complete scorification of the Fe. The furnace was either a Welsh copper smelting furnace, or a furnace built into an iron pan in a similar way as is the modified English cupelling furnace.

Details of reverberatory copper matting furnaces need not be given, as they have been discussed elsewhere.[1] A furnace with hearth 10 to 12 ft. wide, 18 ft. long, and 12 to 15 in. deep, used to treat 20 tons charge in 24 hr.

A furnace of the second class, of comparatively small dimensions, is shown in Figs. 388–392. It is a combination of the English cupelling furnace with its fixed roof, *a*, and movable hearth, *b*, and the reverberatory furnace for concentrating matte, the flame passing from the fire-box, *c*, along the major axis of the hearth to a flue, *d*, placed over the working-door, *e*. A new feature lately introduced in several cupelling furnaces is that the side-walls, *f*, rest on I-beams, *g*, supported by pillars, *g'*, thus leaving the entire bottom of the hearth and test-carriage exposed to the cooling action of the air. The hearth, 6 by 8 ft., is inclosed by water-jackets, *h* and *h'*, and is supported by four screws, *i*,

[1] Hofman, H. O., "Metallurgy of Copper," McGraw-Hill Book Co., New York, 1918, 236.

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with threaded sockets, *j*, in test-carriage, *k*, running on rails. The two front screws can be rotated together by a pair of worm-wheels, *l*, on shaft, *m*, revolved by bevel-wheel and pinion, and these by hand-wheel, *n*; in a similar way the two back-screws can be raised and lowered from the side by turning a crank (not shown). The steel water-jacket, *h*, receives its cooling water at the back (not shown), and is drawn off at the front from the top through pipes, *o* and *o'*; the cast-iron breast-jacket, *h'*, receives its water at *p* and discharges through two pipes, *q* and *q'*; thus the jackets are always filled with water. Rubber hose connects the jackets with the supply- and discharge-pipes. The hearth material is a mixture of raw and burnt clay or of raw clay and quartz. A chromite lining,[1] or better, one of chrome brick, lasts a long time.

The fire-box has a shaking grate, worked with underwind. On either side of the bridge is a tuyère-pipe through which blast enters upon the hearth. The furnace is incased in iron plates and is strongly bound with iron rods. It will bring forward in 3 days about 10 tons of 40-per cent. copper matte to blister copper, the matte being charged in four portions, first 45 per cent., then 25 per cent., 22 per cent., and 8 per cent. The 10 tons of matte produce 8700 lb. blister copper with 95 per cent. Cu and 8300 lb. of slag assaying 13.8 per cent. Cu and 52.3 per cent. Pb, and require 900 gal. of reduced oil with air as atomizer. The furnace is well adapted for concentrating retort bullion if the hearth is made thinner and shallower.

A special mode of operating is the one of Freiberg, Saxony,[2] where matte with 50 per cent. Cu^3 is roasted in a hand reverberatory furnace and then treated in a smelting reverberatory with barite and quartz as fluxes. The hearth of the furnace is 13 ft. long, 4 ft. 2 in. wide at the bridge, 8 ft. at the middle, and 1 ft. 2 in. at the flue; the grate, 4 ft. 3 in. by 3 ft. 4 in., is 1 ft. 3 in. below the top of the bridge, which is 1 ft. 3 in. below the roof. The furnace has two doors, one at the side and one beneath the flue. The bottom, which is built on an iron plate, consists first of a full course of fire-brick on which is rammed firmly a mixture of fire-clay and quartz, giving it the usual dish-like form, with the lowest point at the tap-hole. The thinnest part of this layer is 3 in. thick. The working bottom, which follows, consists of 5500 lb. of an intimate mixture of 5 parts of quartz and 1 part of matte-slag. It is melted down, after being heated and patted into shape, in 12 hr., and lasts from 15 to 18 months. Five charges, each weighing 4290 lb., and consisting of 2640 lb. roasted matte, 440 lb. raw matte, 660 lb. barite, and 550 lb. quartz, are treated in this furnace in 24 hr., 4¼ tons of bituminous coal being consumed. From these five charges 8250 lb. of matte is produced. The composition of matte and slag as analyzed by Schertel[4] is given in Table 93:

[1] Lang, *Eng. Min. J.*, 1897, LXIII, 207.
[2] Grand, *Ann. Min.*, 1875, VII, 314.
Capacci, *Rev. Univ. Min.*, 1881, IX, 269.
[3] Schertel, *Berg. Hüttenm. Z.*, 1888, XLVII, 442.
[4] *Loc. cit.*

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TABLE 93.—ANALYSES OF MATTE AND SLAG FROM FREIBERG CONCENTRATION SMELTING

Matte		Slag	
Pb	4.85	SiO_2	30.60
Ag	0.31	PbO	14.51
Cu	73.95	CuO	4.40
Bi	0.02	FeO	8.10
Sb	0.06	(NiCo)O	0.38
As	0.18	ZnO	1.95
Fe	0.13	Al_2O_3	1.80
NiCo	0.21	BaO	28.63
Zn	.....	CaO	7.70
S	18.98	MgO	0.72
		S	0.18

In order to obtain matte running so low in iron, it is necessary to produce some bottoms. These go back to the next reverberatory charge or are added to the last blast furnace charge to be again converted into matte. The matte forms the raw material for the manufacture of blue vitriol.

The behavior of $BaSO_4$ has been treated in §143. Schweder[1] found in his experiments that $CaSO_4$ and Na_2SO_4 behaved similarly to $BaSO_4$.

Another method of bringing forward matte with 40 per cent. Cu to blister copper is the one in operation at Hoboken-les-Anvers, patented by C. Heberlein[2] where the blast furnace matte is blast roasted in a Huntington-Heberlein pot with the addition of limestone and quartzite, and the product smelted in the reverberatory furnace for blister copper. During the blast roast a large part of the As and Sb is volatilized.

Laboratory experiments by Hesse[3] and Menzel[4] throw light upon the chemical behavior of matte components and fluxes when subjected to elevated temperatures in carbon and electric furnaces.

185. Converting of Lead-Copper Matte, General.[5]—In this process, which is the latest improvement in the treatment of lead-copper mattes, air is forced in small streams through one side of an iron vessel, lined with refractory material, into molten matte held at a temperature of 1100 to 1200° C., with the object oxidizing Fe, Pb, and S, and recovering Cu as metal. The oxidized Fe is slagged; the oxidized Pb is in part slagged, in part volatilized; the S forms SO_2 and passes off. Slagged Fe and Pb are subsequently retreated in the blast furnace, as is the volatilized Pb.

The first successful converting of concentrated lead-copper matte was that at Aguas Calientes early in 1900 using an upright acid-lined vessel. The idea held

[1] *Berg. Hüttenm. Z.*, 1879, XXXVIII, 38.

[2] *Metallurgie*, 1910, VII, 285.

[3] *Metallurgie*, 1911, VIII, 321.

[4] *Metall. u. Erz.*, 1913, X, 193, 230; *Eng. Min. J.*, 1913, XCV, 902; *Abhandlungen Instit. Metallhüttenwesen u. Electrometallurgie*, 1915, II, heft 1.

[5] Vail, *Eng. Min. J.*, 1910, LXXXIX, 503.
Moore, *op. cit.*, 1910, XC, 263.
Kuchs, *Tr. A. I. M. E.*, 1914, XLIX, 579.

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previously[1] that the losses in Ag were enormous was based on a misinterpretation of data coming from A. Raht who held that idea about 1895,[2] but was convinced later on[3] of the contrary, when he found that true fume from the converter assayed only 2 oz. Ag per ton, and that the loss in Ag was proportional to that of Cu; in other words, that the loss was a mechanical and not a volatilization loss. This fact is brought out by the later data of Aguas Calientes given by Semple[4] and shown in Table 94. The converters are upright (diameter, 8 ft.; height from tuyère to base of hood, 7 ft. 4 in.; blast-pressure, 12–16, sometimes 18 lb. per sq. in.); the hoods end in a steel flue having discharge-doors for dust; the gases pass from the flue to an exhaust fan which forces them through four brick sections, a curtained chamber, and four steel cooling flues into the bag house.

TABLE 94.—ANALYSES OF CONVERTER FUME FROM AGUAS CALIENTES

	Weight, pounds	Ag, ounces per ton	Au, ounces per ton	Pb, per cent.	Cu, per cent.	As, per cent.	SiO_2, per cent.	Fe, per cent.	Zn, per cent.	S, per cent.
At throat of converter	.....	126.23	0.513	16.8	34.0	.....	.....	.....	.....	.....
Converter flue-dust	13,966	103.95	0.467	25.2	30.7	.....	0.4	0.5	7.3	12.1
Oval pipe hopper	1,536	86.39	0.327	30.0	24.9	.....	0.6	Trace	8.0	11.1
Fan house, section 1	2,660	58.13	0.175	37.0	16.2	.....	0.6	0.3	9.3	10.7
Fan house, section 2	5,188	78.46	0.321	29.5	20.7	.....	0.7	0.3	8.8	11.4
Fan house, section 3	29,361	16.10	0.029	52.5	0.75	.....	0.4	.....	14.4	8.4
Fan house, section 4	12,287	12.69	Trace	57.0	0.50	.....	0.4	Trace	11.5	8.3
Curtain chamber	28,316	14.44	Trace	54.8	0.90	.....	0.5	Trace	11.1	8.7
Condenser	72,125	13.27	Trace	53.5	0.15	3.6	0.4	.....	10.4	8.9
Bag house 1	36,920	1.18	Trace	58.0	0.10	Trace	0.3	.....	9.8	8.7
Bag house header	2,636	14.73	Trace	58.5	0.10	Trace	0.2	.....	9.4	8.8

The success at Aguas Calientes led to the introduction of the acid-lined upright vessel at Omaha, Neb., in 1905. This was 7 by 14 ft. at the start, it was charged with 3 tons of 40-per cent. matte, in two or three successive additions, and blown to blister copper, amounting to 2500 lb. The second matte-charge gave 5000 lb. copper, the third 10,000, and the fourth 15,000 lb., when the converter had to be relined. The acid vessel was supplemented in 1908 by one with a basic lining. There were in operation in 1912[5] two upright acid and one horizontal basic (Peirce-Smith) converters. As in converting lead-copper matte, the basic vessel has the same great advantages over the acid as in converting lead-free copper-iron matte,[6] basic vessels have entirely replaced the acid.

[1] Peters, E. D., "Modern Copper Smelting," McGraw-Hill Book Co., 1895, p. 569.
[2] Hamilton, *Tr. A. I. M. E.*, 1913, XLVI, 475.
[3] Private communication, March 4, 1911.
[4] *Eng. Min. J.*, 1911, XCI, 508.
[5] Eilers, *Tr. A. I. M. E.*, 1912, XLIV, 708.
[6] Hofman, "Metallurgy of Copper," 1918, p. 334.

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The following discussion is confined to basic horizontal vessels. Here there have to be distinguished two methods of converting. One uses silicious fluxes, as is common in treating lead-free copper-iron matte, and is in operation at Maurer, N. J.; Omaha, Neb.; and perhaps other places; the other works without silicious flux, and is in operation at Tooele, Utah.

All plants have horizontal converters. Recent practice in converting copper-iron matte shows a tendency to replace the horizontal by the vertical; whether the smelteries treating lead-copper matte will follow suit, remains to be seen.

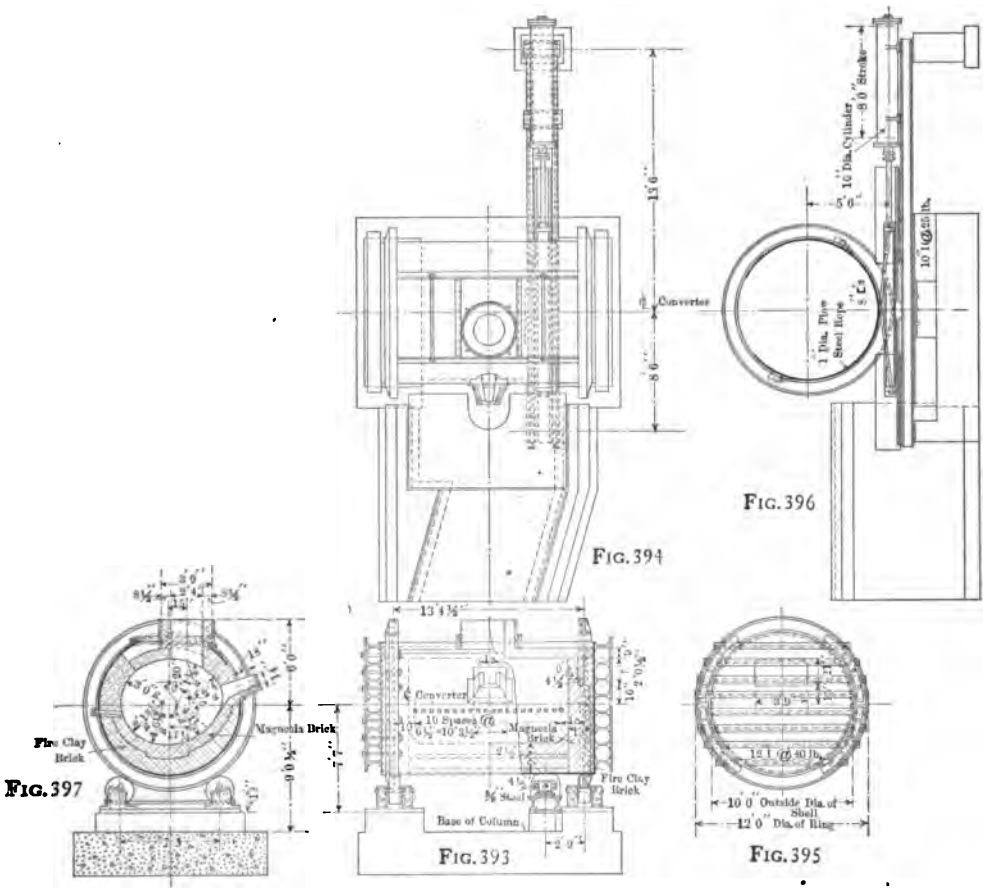

FIG. 394 FIG. 396 FIG. 397 FIG. 393 FIG. 395

FIGS. 393 to 397.—Peirce-Smith converter.

186. Converting with Addition of Silicious Flux; Peirce-Smith Vessel.—The converter, shown in Figs. 393–397, is a horizontal cylindrical lined shell of ⅝-in. steel plate, 14 ft. 10½ in. long and 10 ft. in diameter, holding about 25 tons of copper; it is manipulated by means of a hydraulic piston. The shell is open at the top for about 4 ft., to allow for expansion of the lining, and is tied by heavy iron rods. It has riveted to it two cast-steel annular I-beams forming the treads with which it revolves with two pairs of carrying rollers. The heads

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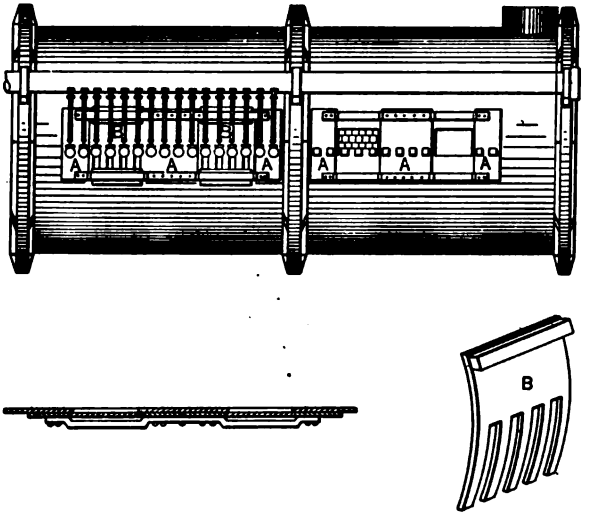
of the cylinder, also of ⅝-in. steel, are not firmly connected, but have flanges which extend into the cylinder in order that the heads may yield to the longitudinal thrust of the expanding lining. They are backed by I-beams which are tied by bolts and nuts to the cast-steel annular treads at the ends of the cylinder. A bolt passes through the annular tread and a piece of flat-iron, which bears upon the I-beam, and is locked with a nut. Loosening of nuts furnishes the space necessary for expansion when the furnace is being heated; bending of the flat-iron indicates the strain put upon the bolt.

The cylinder has 4½ in. beneath the median horizontal plane 20 cast-iron tuyères, 2 in. square, with 1.25-in. holes which extend through the lining and have a downward pitch. The pipes are provided with Dyblie valves[1] and are connected by wired hose to a wrought-iron blast-pipe bolted to the cylinder. The blast-pipe is connected by a cast-iron pipe and swing-joint with the main of the blowing-engine.



FIGS. 398 to 400.—Smith device for repairing tuyère-belt.

Since in converting lead-copper matte, the tuyère-zone, 9 in. above and below the tuyère-pipes, is likely to be quickly corroded, the shell is provided there with movable plates. The Smith device for repairing the tuyère-belt[2] is shown in Figs. 398–400. The shell has oblong openings which are closed in part by movable plates, *A*, bolted on through movable straps; these are secured so as to leave room for the plates, *B*, which are inserted and keyed. The plates are forked at the lower ends that they may pass over the protruding cast-iron tuyères.

Opposite the tuyères, on the breast or pouring side, is the pouring opening with spout. The lower part is closed with a magnesite brick provided with taphole for casting blister copper; the upper with clay in which is left an opening

[1] Hofman, "Metallurgy of Copper," 1918, p. 306.

[2] U. S. Patent No. 1044587, Nov. 19, 1912; *Eng. Min. J.*, 1913, XCV, 322.

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of about 3 in. in diameter for pouring slag, which is collected in a ladle or may be granulated as shown in Fig. 394.

In one end of the vessel is an opening for introducing an oil-burner used for warming and heating a new converter, or for keeping it hot when temporarily out of commission.

In the center of the top is the throat for receiving charges and delivering gases to a lined hood leading into a dust chamber, whence they pass to the condensing apparatus for collecting fumes. The distance between the outer diameter of the nose of the converter and the inner diameter of the hood is about 8 in., in order that sufficient space may be provided for air to enter and cool the gases.

The converter is rotated, Fig. 396, by means of two wire ropes, of which one end is attached to the converter, the other to blocks sliding in guides direct-connected with the piston-rod of the hydraulic cylinder.

The lining is mainly magnesite brick. In order to save in cost of construction, there is placed on the bottom a course and a half of fire-brick along the center line; the thickness of 13 in. tapers on one side to a full course at 8 in. below the tuyère-level, and to a half-course on the other. The working-lining is 9 in. thick except at the tuyères, where it is 18 in. The oval cavity has a length of 11 ft. 4 in. and a width of 7 ft. at the median cross-line. The magnesite brick are laid dry with the spaces filled in with magnesite powder; at the tuyère-belt a mortar of magnesite powder and linseed oil is used. In order to furnish room for expansion, pine shingles, ¼ in. thick, are introduced after every eighth course of 9-in. brick.

Following the example of Mathewson at Anaconda[1] the cast iron of the tuyère-pipes has been replaced in some instances by wrought iron, and the lining of magnesite brick at the tuyère-belt by ramming in place, 8 in. above and below the tuyères, a mass consisting of finely ground magnesite, asbestos, and 20 per cent. of a 40-per cent. solution of sodium silicate.

The blast pressure required is about 15 lb. per sq. in.; 18 tuyères, 1.25 in. in diameter, are used, the two end-tuyères being closed.

The accessories are similar to those used in converting copper matte[2] excepting that the slag may be granulated instead of being cast, and the casting of copper is usually more simple on account of the smaller amounts produced.

The matte, received from the fore-hearth of the blast furnace, is usually stored in a reverberatory furnace, similar to the one shown in Figs. 252 and 254, whence it is tapped into a ladle holding 6 or more tons, and poured into the converter. It can be melted in the converter, but this is exceptional.

The silicious ore, crushed to ⅝ in. and frequently warmed in a mechanical reverberatory, is raised in suitable amounts in a sheet-iron box or boat, and shot into the converter.

187. Process and Manipulation.—The process and mode of operating are

[1] Hofman, "Metallurgy of Copper," 1918, p. 345.

[2] *Ibid.*, p. 314.

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the same as in converting copper-iron matte in so far as the principles involved are concerned; they differ, however, as to detail.

A converter is warmed in about 3 or 4 days by firing with cord wood, and then brought to the desired converting temperature in about 8 hr. by means of an oil flame, or in about 4 hr. by burning 1 ton of coke. The hot converter receives 20 tons of liquid matte (Cu 46, Pb 17, Fe 12, S 19, Ni, Zn, As, etc. 4 per cent.) and then 1 ton of warmed silicious ore; blast of about 5 lb. pressure is admitted; the converter is turned sufficiently to bury the tuyères in the matte, when the bubbling of the bath will cause the ore to spread; it is then turned over farther that the tuyères may be 24 in. below the surface of the matte, which requires increase of volume of air that its pressure may rise to 15 lb. The blowing on matte for white metal lasts about 30 min. It represents the slagging stage in which Fe, Pb, and S are oxidized; the FeO and PbO, as soon as formed, combine with the silicious ore and form slag. The rate of scorification of the ore is recognized by the fluidity of the slag. This is tested by quickly inserting an iron rod into the boiling bath through the slag-pouring opening and withdrawing it. The rod ought to pass readily through the slag and ought to be covered, when withdrawn, with a uniform coating of slag free from unmelted particles. A lack of flux, or a low temperature, or an insufficiency of time of blowing tend to form a mushy slag. When the ore-charge has been satisfactorily slagged, the slag is poured, and 10 tons of matte are poured into the vessel. The converter is righted so that the tuyères become uncovered, the blast is shut off, and about 1000 lb. ore shot in. The blast is again admitted, the converter turned over as before, and blowing continued for about 20 min. The operations are continued, four in all, with decreasing amounts of matte and ore until the converter holds from 28 to 30 tons white metal, which reaches with the righted converter to about 3 in. below the tuyère-level.

The rate at which Fe and Pb are scorified, and the S is oxidized, is similar to that in working copper-iron matte.[1]

There follows the blister-forming stage, in which the 28 to 30 tons white metal are blown, in roughly 1 hr., to about 25 tons of blister copper which assays Cu 92–94, Ag 1±, Pb 1–2, As 1, SbNi, etc. 4 per cent. The converter is turned over and blown, but without ore. The Cu_2S is oxidized to Cu_2O; this reacts upon still undecomposed Cu_2S, and gives Cu and SO_2, as is shown by $Cu_2S + 3O = Cu_2O + SO_2$; $Cu_2S + 2Cu_2O = 6Cu + SO_2$; some Cu_2O enters the small amount of slag formed, or it remains uncombined in the vessel.

A pour of 25 tons blister copper is obtained by blowing 50 tons 50-per cent. matte for about 6 hr., of which 2 hr. are taken up by handling materials.

The progress of a blow is watched mainly by the fume passing off at the throat of the vessel; the color of the flame forms no satisfactory guide, as it is obscured by the fume and vitiated by the impurities of the matte. The end of a blow is ascertained by the color of the slag, which should not be reddish, and by punching a tuyère and quickly withdrawing the bar. This should be

[1] Hofman, "Copper," p. 326.

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covered with a thin skin of copper, rose color and full of pin-holes, but free from dark spots, which indicate undecomposed matte.

During a blow the tuyères have to be punched consecutively. As the work must not be interrupted, two men are required for it on a shift; one furnaceman at the front attends to the running of the vessel. Materials are handled by a separate crew.

The temperature of the charge should not exceed 1200° C., as otherwise the lining is quickly corroded. A low temperature makes the slag mushy. The slag, of which about 1.6 tons is made per ton of blister copper, analyzes SiO_2 25, Fe 25, Pb 20, Cu 10. It is essential that the SiO_2-content be low. A high-SiO_2 slag attacks the lining, and a slag low in SiO_2 causes some Fe of the matte to be oxidized to Fe_3O_4. This adheres to the lining and protects it from attack. Temperature and SiO_2-content are the two leading factors which govern the life of the lining; speed of converting is controlled by the amount of air blown into the converter.

The lining at the tuyère-belt has to be repaired usually after 1000 tons of blister copper have been produced from matte with 40 per cent. Cu; it lasts longer with mattes that run low in Pb, Sb, and As; the roof is good for 2000 tons blister copper; the lining opposite the tuyères and on the bottom is hardly attacked at all.

188. Collection of Lead Fume.—One factor still to be considered is the recovery of lead fume and, with it, of precious metal mechanically carried away by fine particles of base metal. Two methods are used for this purpose, filtering with a bag house and condensation by the Cottrell process.

Filtration by means of bags is in operation at Omaha, Neb., and Maurer, N. J. Details of bag house construction are taken up in §208 and following.

At Omaha[1] the gases, 68,000 cu. ft. per min., from the two upright acid-lined and the single Peirce-Smith basic-lined converters[2] are drawn through a suspended sheet-steel flue, Figs. 418 and 419, 6 by 10 ft. in section and 1250 ft. long, coated inside and outside with graphite paint, by means of a fan, 12 ft. in diameter and 6 ft. wide, making 140 r.p.m. and requiring 25 h.p., into the bag house, 102 by 60.8 by 40 ft. which has a cellar-space of 67,000 cu. ft. and a bag-chamber of 174,000 cu. ft.; it contains 940 woolen bags, 18 in. in diameter (56.5 in. in circumference) and 28 ft. long, which represent a filtering area of 124,000 sq. ft. The house is divided by a vertical wall into two compartments so that one can be cut out when the bags are to be shaken, which occurs twice a day. The cellar is 10 ft. high. The gases pass off into the open through a brick-lined steel stack, 300 ft. high. The life of the bags, which is about 100 days, is governed by the temperature and the SO_3-content of the gases, and by the pressure inside and the vacuum outside.

The following is a record of temperature of converter gases with outside air at 27° C.: back of converter, 375°; at 80 ft. from converter, 292°; at 200 ft., 240°; at 400 ft., 199°; at 600 ft., 174°; at 800 ft., 149°; at 1100 ft., 130°; at fan,

[1] Eilers, *Tr. A. I. M. E.*, 1912, XLIV, 708.

[2] The acid have been replaced since by basic converters.

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119°; at bag house, 67°; at stack, 58° C.; the temperature in the bag house varied from 37° to 77°, and averaged 60° C.

Determinations of SO_2 and SO_3 in the gases are given in Table 95.

TABLE 95.—SO_2- and SO_3-CONTENT OF CONVERTER-GAS

SO_2-content, per cent.				SO_3-content, per cent.	
At converter	At fan	In bag house		At converter	At fan
		Top	Bottom		
3.62	2.05	2.06	2.06	0.038	0.020
2.34	1.21	1.30	1.60	0.027	0.025
4.46	2.68	1.52	1.40	.....	.....

With a vacuum at the converter of 0.1 in. water, at the fan of 0.3 in., the pressure in the cellar is 0.56 in. and higher, reaching 1 to 2 in. in the bags; with a vacuum outside of the bags as low as 0.07 in. that at the base of the stack is 0.3 in.

The matte blown averages: Pb 26.9, Cu 43.1, NiCo 0.04, Fe 8.0, Zn 2.5, S 15.5, As 1.7, Sb 0.76, SiO_2 0.3 per cent., and Au 0.87, Ag 98.1 oz. per ton. The blowing engine furnishes 5800 cu. ft. air per min. at a pressure of 17 lb. per sq. in. With a daily output of 45 tons blister copper from matte with Pb 23, Cu 45, S 18 per cent., there are produced 16 tons of bag house fume with Pb 66.0, Zn 1.9, Sb 0.2, Te 0.2, Se 0.24, combined SO_3 21.63, free SO_3 0.2 per cent., and Ag 2.8 oz. per ton. The composition of the dust collected in the flues is given in part in Table 96.

TABLE 96.—DUST OF CONVERTER FLUE

Distance from converter, feet	Pb, per cent.	Zn, per cent.	SO_3, per cent.		Ag, oz. per ton
			Combined	Free	
200	56.0	1.6	25.13	0.25	4.2
400	60.0	1.6	23.13	0.55	3.0
600	64.2	1.3	22.82	0.15	2.4
800	64.6	1.3	22.60	None	2.0
1,100	63.0	1.5	23.33	None	2.1
1,200	64.4	1.6	20.60	None	2.6

The data show a general fall of Ag- and rise of Pb-content with increase of distance from converter to bag house, and a reduction of free SO_3 which disappears at 800 ft. from the converter. It brings out the important influence the presence of lead fume has upon the neutralization of free SO_3. The gases from matte containing less than 14 per cent. Pb cannot be filtered without the admixture of other neutralizing agents such as CaO or ZnO as in the Sprague process (§212).

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In the first attempt to filter converter gases at Maurer, N. J.[1] the bags were corroded in 30 min. during the finishing blow for blister copper, when the Pb-content of the matte had been reduced to 2 per cent. The throat of the converter had been fitted closely into the brick flue leading to the bag house. The temperature of the flue back of the converter ranged from 426° to 704° C. and at the opposite end from 204° to 371°, thus covering a range well suited to convert SO_2 into SO_3, when the SO_3 not combined with PbO attacked the filtering-fabric. Reduction of temperature to a maximum of 315° C. in the brick flue 25 ft. from the converter, and diminution of the SO_2-content of the gases, from as much as 9 per cent. vol. to 4 per cent., corrected the evil to such an extent that even cotton bags lasted over 1 year. It was accomplished, by enlarging the brick flue opposite the converter so that air would enter with the sulphurous gas, by a dampered port in the flue through which additional air could be drawn in, and by speeding up the fan.

The Cottrell process for the condensation of converter fume was first introduced at the copper smeltery of Garfield, Utah, by Howard,[2] and has since been installed at other plants.

A discussion of the process is given in §214.

The matte produced at Garfield in the blast furnace contains from 20 to 40 per cent. Cu, that in the reverberatory furnace from 33 to 48 per cent.; and more or less Pb beside the usual amounts of Fe and S. It is blown in basic-lined Peirce-Smith converters (cylinders 24 ft. long and 10 ft. in diameter) with a pressure of 10 lb. per sq. in. through 37 tuyères 1¼ in. in diameter, placed 18 in. above the bottom. The construction of the original converter has been changed[3] in order to reduce the amount of air entering the flue with the converter gas, and thus to diminish the volume of diluted gas which is to pass through the electric precipitator.

There are in operation seven converters which furnish per min. 250,000 cu. ft. gas at flue temperature, equal to about one-half this volume at standard conditions. The gas travels through an elevated balloon flue, 1000 ft. long and 210 sq. ft. in cross-section, at a speed of about 12 ft. per sec. Most of the Cu-bearing dust settles in the flue; the gas is cooled to about 90° C. before it enters the electric treater. This is shown in Figs. 401–403. It contains seven units, one for a converter; generally six are in operation, and one is being cleaned or repaired. A unit has 360 grounded vertical cast-iron pipes, 5 in. in diameter and 10 ft. long, serving as precipitating electrodes. In the center of each is held taut a steel wire of No. 14 gauge which, serving as discharge electrode, is connected with the intermittent direct current of 30,000 volts. There is, therefore, maintained in each pipe an electrically charged field through which the gases, cooled to below 100° C., travel and deposit on the pipes all dust, fume, and moisture. The deposit, which does not drop off, is loosened by knockers at intervals into the lower gas chamber, provided with openings,

[1] Alexander, *Tr. A. I. M. E.*, 1914, XLIX, 561.
[2] *Tr. A. I. M. E.*, 1914, XLIX, 540.
[3] *Met. Chem. Eng.*, 1915, XIII, 924.

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closed by gates, to remove the dust. The discharge-wires have similar knockers to detach adhering particles of fume. The gases, in which the ratio of $SO_3 : SO_2$ is 1:18, are admitted from the converter flue to a unit through three

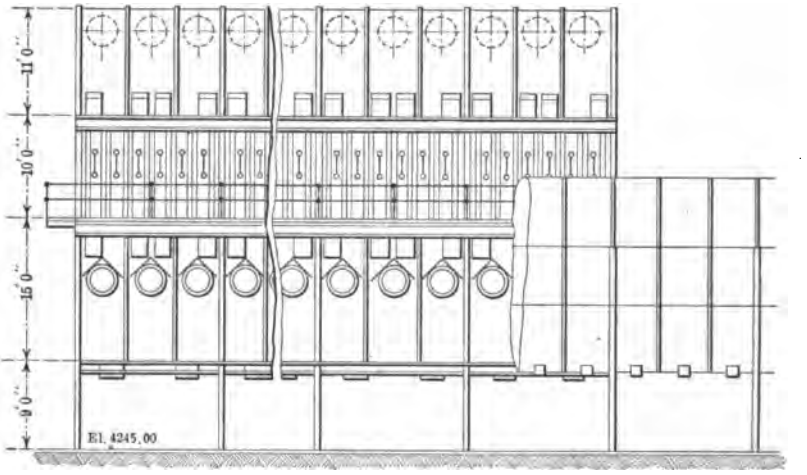

FIG. 401.—Longitudinal elevation.
Cottrell plant, Garfield, Utah.

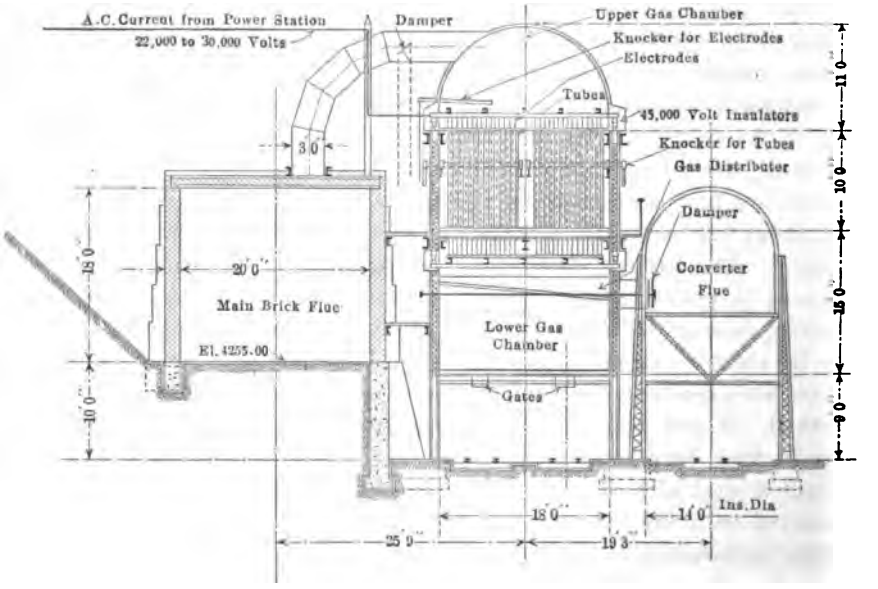

FIG. 402.—Cross-section.
Cottrell plant, Garfield, Utah.

pipes, provided with dampers, into the lower gas chamber. This has three compartments in order to attain an even distribution of gas. Each pipe has a gable-shaped roof which serves a similar purpose. The treated gas of a

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section leaves the treatment chamber through three pipes, provided with dampers, and enters the main brick flue of the whole plant on the opposite side, which is 18 ft. high, 20 ft. wide, and ends in a stack 350+85=435 ft. high and 22 ft. inner diameter at the top. An analysis of the collected dust showed SiO_2 1.57, Insol. 0.39, PbO 55.02, CuO 6.75, Fe_2O_3 1.82, SO_3 24.50, CaO 0.50, As_2O_5 5.21, Bi_2O_3 1.91, H_2O 2.08, Ag 0.02, total 99.77 per cent. The recovery of Pb is 97.25 per cent.

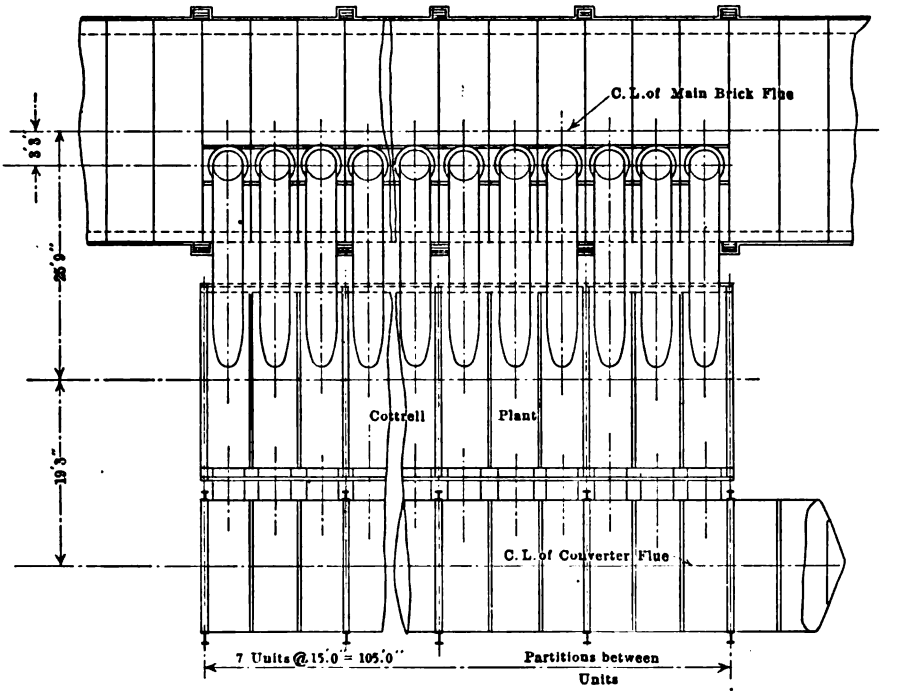

FIG. 403.—Plan, Cottrell plant, Garfield, Utah.

The electrical apparatus contains seven complete motor-generator rectifier sets and transformers, and a switch-board panel. A set consists of a 30-h.p. 250-volt D.C. motor, direct-connected with a 20-kw., 4-pole, 60-cycle, 220-volt, single-phase A.C. generator. The 220-volt current is stepped up to 20,000 to 30,000 volts by means of a 20-kw. transformer, and then led back to the rectifier which is on the same shaft as the motor generator.

In the preliminary experimental work it was found that in the slagging period, when the gases contain little SO_3, the pipes became covered with a conductive coating which stopped the work. Injecting moisture in sufficient amount to obtain a dust with 3 to 5 per cent. H_2O corrected the evil. During the blister period there is enough SO_3 in the gas for satisfactory work. It was also noticed that, though at 340° C. lead was precipitated, no SO_3 or As_2O_3 was

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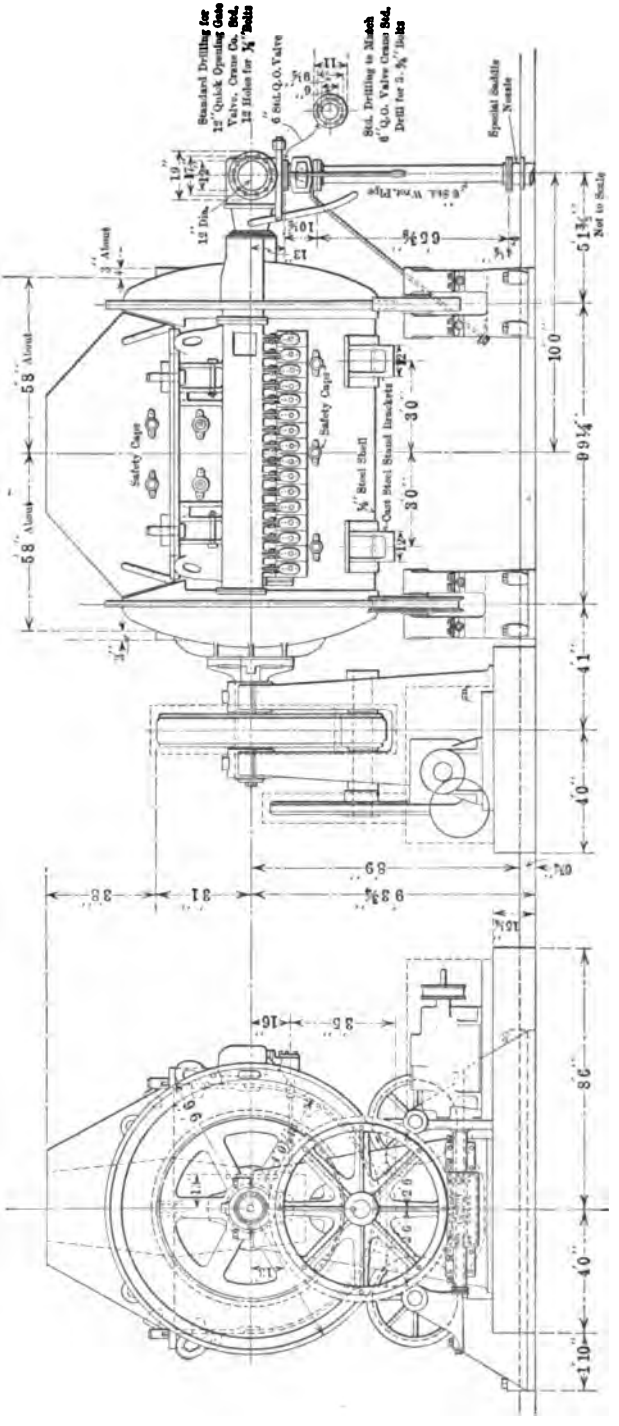

FIG. 404.—Side elevation.

FIG. 405.—End elevation.

Basic barrel converter, Tooele, Utah, 96 × 150 inches.

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deposited. It may be recalled that SO_3 boils at 46° C.; that H_2SO_4 of 60° Bé. is vaporized at 200° C., and As_2O_3 at 143° C.[1]

189. Converting Lead-Copper Matte Without Addition of Flux; Tooele, Utah.[2]—This method of converting had its origin at Tooele, Utah. The basic-lined barrel converter is shown in side- and end-elevations in Figs. 404 and 405, and in longitudinal and transverse sections in Figs. 406 and 407. The barrel, 96 by 150 in., consists of the usual two parts, the boiler-iron shell and hood, which are joined by key-bolts. It rests with steel runners on two pairs of friction rollers carried by a roller-stand. At one end it has a trunnion which carries a spur-wheel driven by a pinion, the shaft of which carries at the opposite end a worm-wheel engaging with a worm attached to the shaft of an electric

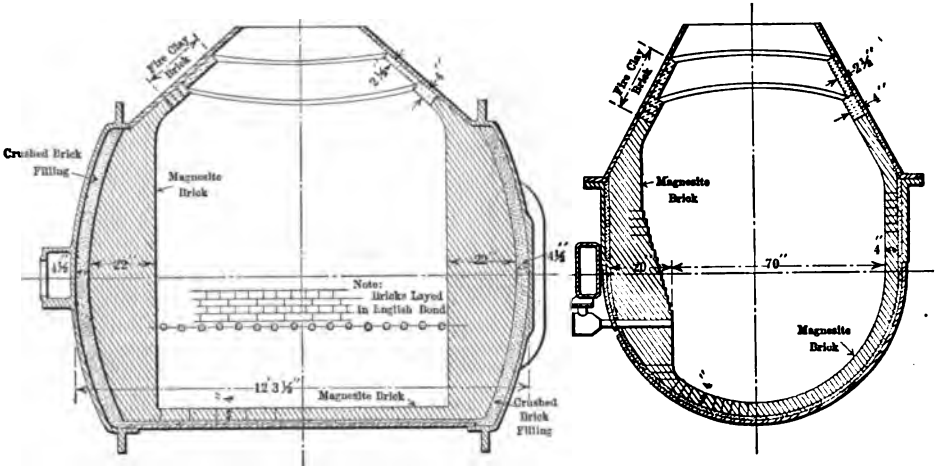

FIG. 406.—Longitudinal section. FIG. 407.—Cross section.
Basic barrel converter, Tooele, Utah, 96 × 150 inches.

motor. At the other end, connection is made with the blast-pipe which delivers the air from the blowing-engine through 16 12-in. tuyères. The shell is lined with standard magnesite brick backed by crushed-brick filling. The thickness of the lining on the tuyère-side of 20 in., and on the ends of 22 in., is in striking contrast with that on the bottom and the side opposite the tuyères, which is 4½ in.

190. Process and Manipulation.—In blowing lead-copper-iron matte, the metallic sulphides are oxidized, but not scorified, owing to the absence of SiO_2. Oxides of lead and zinc are volatilized and collected in a bag house; oxides of copper and iron and some metallic copper form a fluid mass, are poured into a ladle, and transferred to a converter treating lead-free copper-iron matte with silicious ore in the usual manner just after this has received its charge. The oxides of copper are sulphurized, the iron oxides scorified, and the metallic copper taken up by the matte.

[1] Elton, *Tr. A. I. M. E.*, 1913, XLVI, 690.
[2] Kuchs, *Tr. A. I. M. E.*, 1914, XLIX, 579.
Private notes, July, 1916.

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The advantages of the process over the one using silicious flux are, that very little Pb-bearing slag is formed, which would have to be smelted in the lead blast furnace (in the ordinary process 45 per cent. of the Pb in the matte is slagged); and that, on account of the large amount of Pb- and Zn-fume formed, there is present in the gases very little free SO_2 to corrode the woolen bags; in fact, 50 per cent. of the Zn in the matte passes off with the gases, as against 10 per cent. when silicious ore is used.

The plant, shown in Fig. 408, has five converter stands, of which three treat lead-copper-iron matte, and two copper-iron matte. The gases from lead matte converters are drawn by a No. 20 Sirocco fan (capacity 180,000 cu. ft. gas per min.) through a hopper-bottom steel flue, provided with inlet for cold air to reduce the temperature to 100° to 110° C., and delivered into a brick flue with

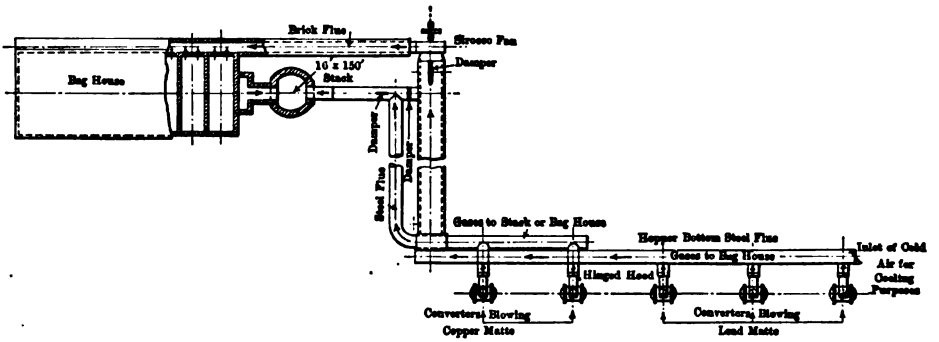

FIG. 408.—Converter flues and baghouse, Tooele, Utah.

branches leading into the bag house; this has 960 woolen bags, 18 in. in diameter and 30 ft. long with 24 threads to the inch.[1] The gases from the copper-matte converters pass through a steel flue to a stack 16 ft. in diameter and 150 ft. in height.

In blowing lead matte, a large volume of dense fumes is evolved at first; the fumes diminish when the blow is about half finished, and disappear gradually. At the same time the flame becomes more and more visible; when plainly visible, the weight of the matte has been reduced to 60 to 70 per cent. of the original charge; the matte retaining 1 to 1.5 per cent. Pb.

A converter receives 9 tons matte, is blown 2 hr. with a pressure of 12 to 13 lb.; the tuyères have to be punched frequently. The time between blows is ½ hr. One skimmer attends three converters, one puncher and helper are required for every vessel. The converter is patched by pouring into the weakened lining Pb-bearing converter slag to the required thickness, which is allowed to come to a set in 1.5 to 2 hr. At the tuyère-belt, closed iron pipes are inserted, and slag poured around them.

The elimination of Pb and Fe is quick at first and less so toward the end of a blow; the oxidation of Zn is slow and more uniform than that of the other components. The temperature in the vessel is not allowed to exceed 930° C.

[1] In a new installation the flues would be made larger and longer.

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Incrustation of the vessel by Fe_3O_4 occurs to some extent on the wall opposite the tuyères, and is heaviest at the mouth of the converter owing to the spattering of the charge, but with a throat of sufficient diameter this evil is not serious. Corrosion at the tuyère-belt is more severe than in blowing lead-free copper-iron matte and necessitates frequent patching. More or less continuous punching of the tuyères is a necessity.

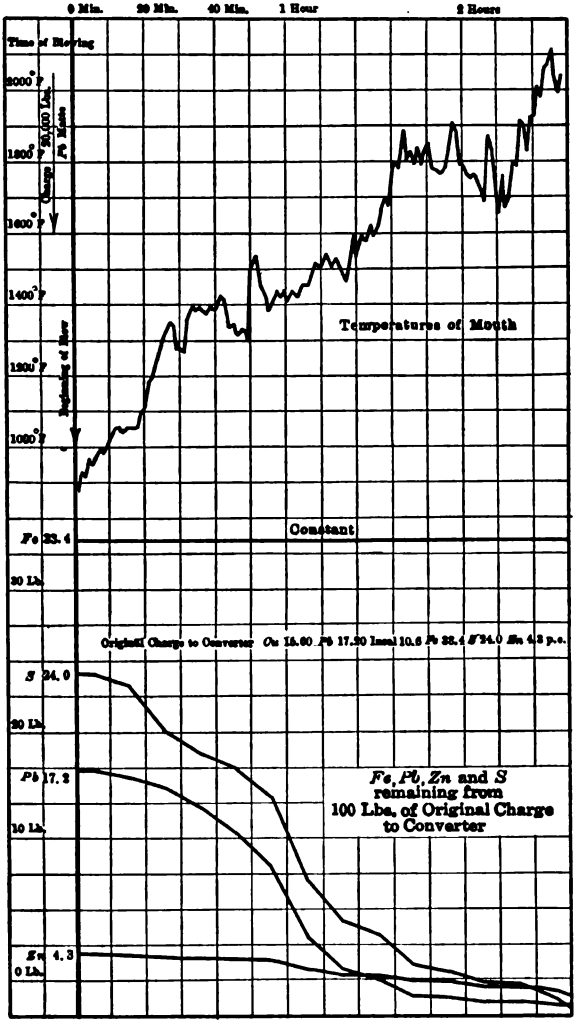

FIG. 409.—Converting lead-copper-iron matte, Tooele, Utah.

Temperature- and elimination-curves are shown in Fig. 409.

Table 97 gives analyses of the raw materials used and of the products obtained in converting lead-bearing and lead-free copper matte. The lead-bearing matte ought not to contain over 7 per cent. Zn, as otherwise mush is formed instead of clean slag.

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TABLE 97.—RAW MATERIALS AND PRODUCTS OF CONVERTING, TOOELE, UTAH

	Pb	Cu	SiO_2	Fe	CaO	S	Zn	Ag*
Blow of Lead-bearing Copper-Iron Matte								
Lead-bearing matte	15.0	9.05	.....	37.9	.....	23.0	5.4	20.3
Transfer-slag to copper converter	1.4	.....	1.7	59.1	.....	18.0	4.0	.....
Bag-house fume	64.2	0.37	0.2	0.4	.....	6.0	10.4	5.7
Blow of Lead-free Copper-Iron Matte								
Lead-free matte	.....	22.2	.....	41.7	.....	26.6	.....	21.8
Silicious ore	3.4	0.3	75.1	7.4	3.0	0.8	.....	17.1
Converter slag	2.3	1.79	23.4	48.8	1.5	1.2	2.9	1.6

* Ounces per ton.

A section of the bag house and the suspension and shaking devices of the bags are shown in Figs. 410–415. The bag house, built of brick, is 125 ft. 6 in. by 47 ft. 6 in. outside, and 50 ft. high to the eaves. It is divided by a sheet-steel thimble-floor into the filter-chamber above and the cellar-space below. There are eight hopper-bottom cellars, 13 ft. 7½ in. high; six are 13 ft. 11 in. by 44 ft., and the two end-cellars are 15 ft. 5 in. by 44 ft. The filter-chamber has 960 woolen bags, or 120 to a cellar, 18 in. in diameter and 30 ft. long. A bag is suspended from a 2½-in. iron pipe by a No. 8 soft-steel wire which passes through a ½-in. pipe 4 in. long forming a loop; the upper end of the bag is gathered, turned over the short pipe, and tied. Each cellar is connected with the main gas flue and the reverse-current gas flue by means of ports to be closed by valves. The gases from the converters arrive in the main gas flue, enter the cellars, pass through the thimble-floor openings into the bags, leave the filter-chamber at the top through a flue ending in a brick down-take which leads to a brick stack, 15 by 150 ft. During this time the ports in the reverse-current gas flue are closed. When the dust inside the bags of a cellar is to be dislodged, the port in the main gas flue is closed and that in the return-flue (joined to a No. 4½ Sirocco fan which delivers into the main flue) opened. The pressure in the bags is thereby replaced by a slight vacuum, causing the bags to collapse and to shake off adhering dust. By repeating the swelling and collapsing of bags a few times every 8 hr., the pores of the filter cloth are kept open. As this reverse-current bag-shaking system is not quite as effective as shaking by hand or mechanically, there is provided an auxiliary shaking device consisting of horizontal chains, each enclosing the suspension wires of a row of bags and having a strong spiral spring in the middle and handles at the ends. Pulling a

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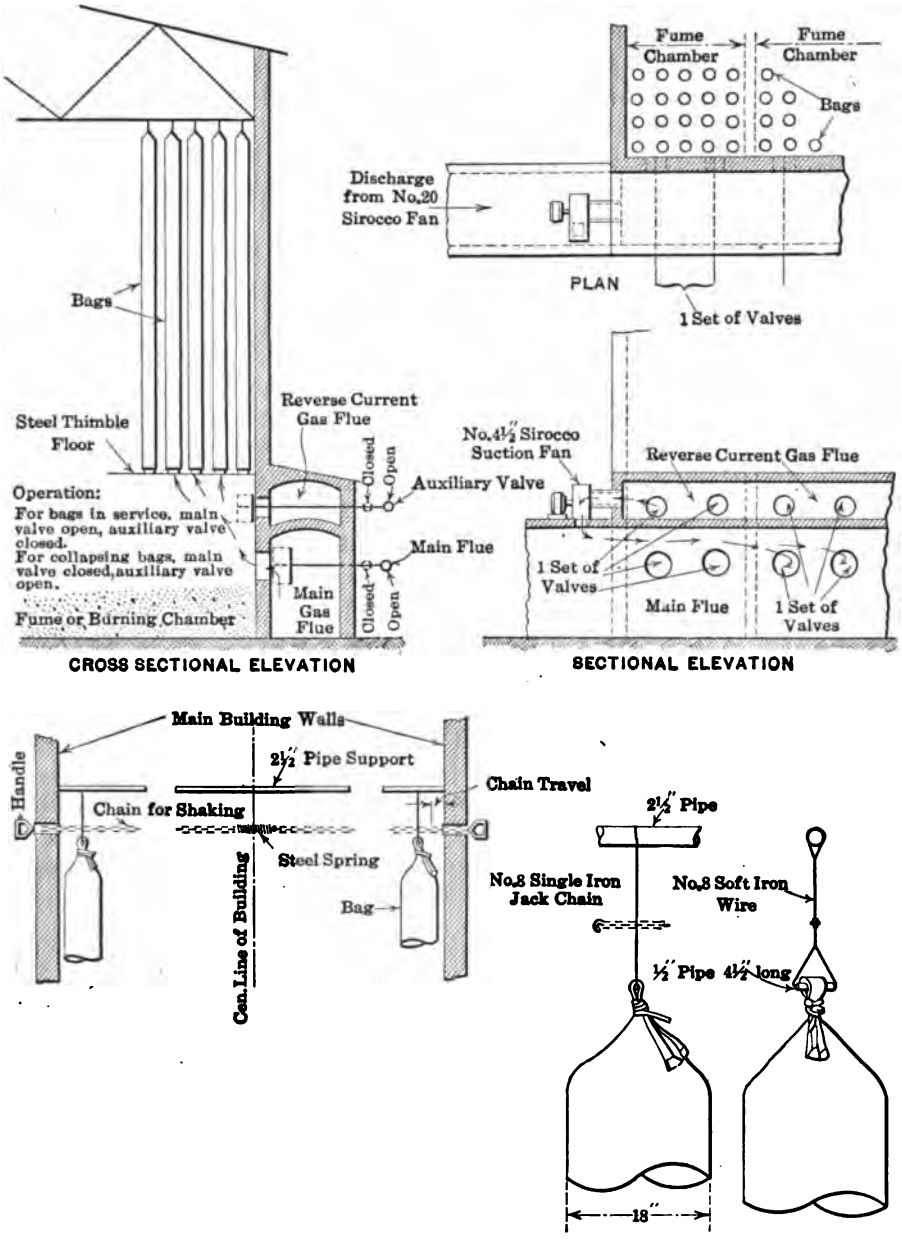
chain and suddenly releasing it, causes the bag to receive a vibratory motion from top to bottom. The dust accumulating in the cellars is removed by a system of



FIGS. 410 to 415.—Reverse-current baghouse and shaking device, Tooele, Utah.

screw and belt conveyors discharging into a car, is then pugged, and added to the blast furnace charges.

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The loss of Pb in converting is from 3 to 5 per cent., that of Ag is negligible.

191. Slag.—The composition (§131–133) and disposal (§117–120) of lead slags have been discussed already.

Waste slag has been used in different ways instead of allowing it to accumulate on the dump. Thus the heat of the slag has been utilized for heating blast[1] and for raising steam;[2] the slag has been cast in the forms of bricks, blocks, tiles, etc.;[3] it is used for ballasting railroads, macadamizing roads; granulated, it has gone into concrete, etc.

Efflorescences on slags have been noticed by Iles-Shelby[4] which they found to consist of K_2SO_4 and Na_2SO_4.

There remains to be considered the cleaning of slags, that is, freeing them of their Ag-content, if this is too high to allow them to go to waste. The Ag-content may be due to entangled particles of lead bullion and especially of matte. Matte may be held in solution by the slag (§131–133); usually, however, it is held in suspension owing to imperfect settling (§118), or to superficial oxidation of matte either by Fe_3O_4 contained in the slag,[5] or by air while tapping:[6] This oxidation sets free SO_2 which, adhering to pellets of matte, hinder them in the settling. Resmelting may recover parts of entangled values but usually not in sufficient amounts to pay for the cost;[7] addition of Cu-bearing pyrite or matte (§147) greatly assists in desilverizing a slag. Thus Keller[8] smelted at Leadville slag from treating carbonate lead ore, estimated to contain 5.3 oz. Ag per ton, with 13 per cent. pyritic ore containing 10 per cent. Cu and 11 oz. Ag per ton in an oblong blast furnace 36 by 80 in. He put through in 24 hr. 113 tons of charge with 10.9 per cent. coke, a little more than half the amount then required to smelt ore-charges. There resulted 5.3 per cent. matte containing 20 per cent. Cu and 92.7 oz. Ag per ton, showing a saving of 80 per cent. of the silver contained in the original slag.

In taking a sample for analysis from a freshly drawn pot of slag, the hardened surface is perforated, and a clean steel bar inserted 3 in. into the liquid slag and quickly chilled by plunging into cold water. Sometimes slag is dipped out from the pot with a clean, cold iron ladle, poured out again after a minute or

[1] Bretherton, *Eng. Min. J.*, 1899, LXVIII, 604, 698; 1900, LXIX, 614, LXX, 760; *Min. Sc. Press*, 1900, LXXXI, 572; 1912, CIV, 243.

Lang, *Eng. Min. J.*, 1896, LXII, 78; *Min. Sc. Press*, 1898, LXXVII, 417; *Eng. Min. J.*, 1909, LXXXVIII, 916.

[2] Giroux, *Min. Sc. Press*, 1893, LXVII, 34.

Howell-Ashcroft, *Eng. Min. J.*, 1893, LVI, 52; 1894, LVIII, 56.

Lang, *op. cit.*, 1909, LXXXVIII, 96.

Göpner-Vautin, *Metallurgie*, 1910, VII, 161.

Johnson, *Met. Chem. Eng.*, 1911, X, 662; *J. Iron and Steel Inst.*, 1914, II, 98.

Perkins-Barker, *Eng. Min. J.*, 1912, XCIII, 267.

[3] Hofman, "General Metallurgy," 1918, p. 469.

[4] *Eng. Min. J.*, 1899, LXVII, 650.

[5] Hering, *Oesterr. Zt. Berg. Hüttenw.*, 1893, XLI, 238.

[6] Vambera, *Oesterr. Jahrb.*, 1903, LI, 135.

[7] Pigott, *Eng. Min. J.*, 1916, CII, 626; see also p. 371.

[8] *Tr. A. I. M. E.*, 1892–93, XXI, 71.

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two, and the thin shell of glassy slag adhering slightly to the ladle taken as sample. The former method, however, is preferable.

192. Wall-Accretions.[1]—These accretions begin in lead furnaces just above the water-jackets and reach up to the feed-door. They sometimes come from the galena in the charge, or from galena formed artificially during the descent of the charge, which adheres to and filters into the brick walls of the furnace; but their principal origin is in the volatilization of lead, zinc, and their compounds, which takes place in the lower parts of the furnace. These fumes condense in the upper cooler parts. During their ascent they are partly oxidized, and the oxides may act chemically on the unaltered parts. Thus in wall accretions may be found volatilized metal, metallic sulphides, arsenides, antimonides with their oxides, and secondary products; sulphides are more apt to be formed near the throat of the furnace and oxides near the top of the jackets. Any insoluble silicates found in the accretions come from parts of the furnace lining or from fine particles of the charge. Analyses are given in Table 98.

TABLE 98.—ANALYSES OF WALL ACCRETIONS

	Lead-ville (*a*)	Lead-ville (*a*)	Lead-ville (*a*)	Pueb-lo (*b*)	Pueb-lo (*b*)	Tarno-witz (sul-phide) (*c*)	Tarno-witz (ox-ide) (*c*)	Claus-thal (*d*)	Denver (*e*)	Den-ver (*f*)	Den-ver (*g*)
S	8.291	2.600	2.725	25.98	19.74	13–9	.....	14.67	10.90	.....	.....
Pb	47.491	70.631	25.953	1.48	37.48	20–32	24	82.71	(*h*) 24.06	.....	.....
PbO	0.405	.....	.....	.....	.....	.....	.....	.....	.....	21.40	.....
Bi	Trace	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Ag	0.0754	0.297	0.0944	0.0188	0.0824	0.005	0.005	.....	(*h*) 29.74	.....	.....
Au	Trace	0.0007	.....	.....	.....	.....	.....	.....	(*h*) 0.197	.....	.....
Cu	Trace	.....	.....	None	None	.....	.....	.....	0.80	.....	.....
Zn	6.977	Trace	53.392	45.68	38.99	30–27	60	Trace	18.42	.....	15.00
ZnO	0.300	.....	.....	.....	.....	C. 2	.....	.....	.....	81.00	.....
Cd	Trace	.....	.....	.....	.....	9–6	.....	.....	.....	.....	.....
Fe	6.754	0.028	.....	6.71	0.94	.....	.....	2.03	13.26	.....	.....
Fe_2O_3	1.100	3.456	1.381	.....	.....	.....	.....	.....	.....	0.40	.....
Mn_2O_3	2.887	.....	.....	0.54 Mn	None	.....	.....	.....	1.60 Mn	.....	.....
As	0.039	5.009	.....	None	None	.....	.....	.....	.....	.....	.....
As_2O_3	.....	.....	0.071	.....	.....	.....	.....	.....	.....	.....	.....
Sb	Trace	2.697	.....	Trace	0.12	.....	.....	1.79	.....	.....	.....
Sb_2O_3	.....	.....	0.056	.....	.....	.....	.....	.....	.....	.....	.....
Sn	.....	5.593	.....	.....	.....	.....	.....	.....	.....	.....	.....
SO_3	0.965	.....	0.287	.....	.....	.....	.....	.....	.....	3.00	.....
P_2O_5	2.166	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Cl, Br	.....	.....	0.035	.....	.....	.....	.....	.....	.....	.....	.....
CO_2	Trace	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
CaO	5.361	.....	0.200	6.20	0.30	.....	.....	.....	3.30	.....	.....
MgO	4.297	.....	0.159	.....	.....	.....	.....	.....	.....	.....	.....
Al_2O_3	1.672	0.600	0.328	.....	0.57	18	.....	.....	.....	.....	.....
SiO_2	10.100	2.830	1.577	8.13	.....	.....	.....	.....	17.26	0.40	.....
O	.....	6.256	13.751	.....	.....	.....	.....	.....	.....	.....	.....

(*a*) Emmons, "Geology and Mining Industry of Leadville," 727. (*b*) Dewey, Bulletin No. 42, "United States National Museum," 1891, 54. (*c*) Dobers and Dziegiecki, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1884, XXXII, 102. (*d*) Mezger, *Berg. Hüttenm. Z.*, 1853, XII, 52. (*e*) Iles, *School Min. Quart.*, 1896–97, XVIII, 18. (*f*) Livingstone, private communication, August, 1896. (*g*) Average for 17 years, Iles. (*h*) Ounces per ton.

[1] Iles, *School Min. Quart.*, 1896–97, XVIII, 18.

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The finer these particles, the more rapidly do accretions form; thus with galena concentrates, accretions form more rapidly than with lump galena. While they generally form at the sides rather than at the ends of a furnace, it has been noticed in some instances that with the increase of the distance between the tuyères they form more at the ends and are thicker at the front than at the back.

The Clausthal analysis, where galena is smelted raw, represents a crystallized wall accretion, consisting principally of galena. At Tarnowitz, where slag-roasted galena rich in blende is smelted with the gray slag from the reverberatory furnaces, two kinds of accretions form, one a sulphide principally black, the other an oxidized compound having a greenish color. The three analyses by Guyard of Leadville accretions (produced when carbonate ores formed, if not the whole, at least the major part of the charge) show a great variety in composition. The lead, for instance, is present as metal, as sulphide, and as oxide; zinc, arsenic, and antimony as sulphides and as oxides.

The method of removing these accretions, while the furnace is running, and their treatment have been discussed in §172.

193. Hearth Accretions or Sows.—In a furnace with an Arents siphon-tap these unwelcome products form on top of the lead below the tuyères; in furnaces tapped from the bottom, they form there. They result from a faulty charge or from a lack of fuel, and are generally mixtures of slag, speiss, matte, metallic iron, metallic lead, coke, and charcoal. The metallic iron results from the reduction of ferric oxide, some of it being held in solution by melted matte and dropped when this cools. The iron of a sow is generally carburized and contains Si and P. Analyses are given in Table 99.

It does not usually pay to work up a hearth accretion; it is thrown over the dump or buried, being an eyesore. Flechner[1] suggests several methods of working furnace-sows. The following, used at the nickel works of Schwerte in Westphalia (Prussia), is of interest. Furnace-sows containing from 75 to 85 per cent. Fe, 5 to 8 per cent. Cu, 3 to 6 per cent. Mo, 2 to 4 per cent. NiCo, and weighing from 500 to 600 lb. apiece, are gradually melted down with coke on the bottom of a blast furnace, the melted parts running out continuously. In this way a large crust is easily reduced in size, and can then be added again to the ore-charge, where it will be taken up by the speiss and the matte. Any mechanical means of breaking up a hearth accretion is sure to cost more than will be recovered from resmelting.

194. Furnace Cleanings and Furnace Refuse.—Furnace cleanings and refuse are a mixture of fire-brick, metal-bearing materials, fuel, etc., obtained in cleaning out a blast furnace when blown down. They are assorted; the waste goes to the slag-heap, the valuable part is added to the ore-bed.

195. Furnace Gases.—The composition of blast furnace gas has been given in Table 75. As it runs high in N, and as the amount of CO_2 it contains is greater than that of CO, it has no value as a gaseous fuel, and goes to waste. The recovery of dust and fume it carries along with it, is taken up in §196.

[1] *Oesterr. Zt. Berg. Hüttenw.*, 1889, xxxvii, 196.

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The gas carries such small amounts of SO_2 and SO_3 that it can pass into the open without having to be treated to neutralize free acids.

TABLE 99.—ANALYSES OF HEARTH ACCRETIONS OR SOWS

	Leadville (*a*)	Přibram (*b*)	Přibram (*b*)
Fe	72.828	48.685	38.875
Pb	18.793	7.309	20.140
Ag	0.1149	0.060	0.160
Au	0.00003	.....	.....
Cu	Trace	0.199	1.107
As	5.083	8.446	2.389
Sb	Trace	0.450	0.735
Mo	0.161	.....	.....
Ni	0.045	Trace	Trace
Co	Trace	Trace	Trace
Zn	Trace	3.610	5.417
Mn	0.015	.....	.....
S	0.650	13.760	18.466
P	0.109	.....	.....
Graphite	0.750	.....	.....
Combined carbon	0.550	.....	.....
Si, slag, loss	0.900	.....	.....
CaO	.....	0.166	0.780
SiO_2	.....	15.850	9.300

(*a*) Emmons, *loc. cit.*, 723. (*b*) Balling, *Berg. Hüttenm. Z.*, 1867, XXVI, 419.

196. Flue- or Chamber-Dust.[1]—This product is the material which has settled from the gases in their passage through condensing flues, chambers, etc. It is made up of two components which merge into one another, dust proper and fume.

The dust consists of particles of the charge, mostly unaltered but sometimes chemically changed, ranging in size from ½ and ¼ in. to impalpable powder; the fume, of metals and their compounds which have been volatilized in the lower part of the furnace and not condensed in the upper. The vapors have been more or less oxidized; so the fume may contain sulphides, sulphates, and oxides of Pb,Zn,As, Sb, Se, Te, etc. The fume is an impalpable powder which collects mainly on the walls of the condensing system; the major part passes off with the gases unless it is arrested by filtration or electric precipitation. Some dust settles with fume, and some fume with the dust. A rough

[1] Hering, C. A., "Verdichtung des Hüttenrauches," Cotta, Stuttgart, 1888.
Guillemain, C., "Theorie und Praxis der Staubverdichtung und der Reinigung und Entstäubung von Gasen," Knapp, Halle, 1911 (mainly a compilation of German patents).
Fulton, C. A., "Metallurgical Smoke," Bureau Mines, *Bull.* 84, Washington, 1915.
Hofman, H. O., "General Metallurgy," McGraw-Hill Book Co., New York, 1918.
Iles, *Eng. Min. J.*, 1886, XLI, 74, 93, 148; *School Min. Quart.*, 1895–96, XVII, 97.
Friedrich, *Metallurgie*, 1906, III, 747.
Ingalls, *Eng. Min. J.*, 1911, XCII, 1115.
Schiffner, *Metall. u. Erz.*, 1914, XI, 257–278.

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distinction can be made by the amount of insoluble residue each contains; dust contains over, fume under 5 per cent. Insol.

Table 100 gives some older analyses of flue-dust from various smelteries,

TABLE 100.—ANALYSES OF FLUE-DUST FROM VARIOUS SMELTERIES

	Wyandotte, Mich. (a)		Pueblo (b)	Harz Mts. (c)	Ems (d)		Freiberg (e)		Sheffield (f)		Tarnowitz (g)
Pb	.....	.....	.....	34.8	60.48	67.04	35.02	27.90	.....	PbS 2.25	.....
PbO	19.91	23.77	37.65	18.0	.....	.....	.....	.....	44.80	68.35	26.4
Pb_2SiO_4	.....	.....	.....	2.9	.....	.....	.....	.....	.....	.....	.....
Zn	.....	.....	.....	1.0	3.17	4.22	5.28	49.50	.....	.....	.....
ZnO	0.09	Trace	5.32	1.5	.....	.....	.....	.....	4.80	1.80	57.4
Cu	.....	.....	Trace	.....	Trace	Trace	Trace	.....	+Bi 1.52*h*	.....	.....
CdO	.....	.....	.....	.....	.....	.....	1.30	.....	.....	.....	.....
As	.....	.....	.....	3.0	0.24	0.16	28.03	1.60	3.03(*h*)	.....	.....
Sb	.....	.....	.....		0.42	0.31	.....	.....		.....	.....
Ag	0.286	0.292	0.04	.....	0.003	0.003	.....	.....	.....	.....	.....
Au	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Fe	.....	.....	.....	1.0	.....	.....	.....	.....	.....	.....	.....
Fe_2O_3	14.43	18.54	24.98	4.5	2.12	1.00	1.57	.....	Trace	.....	.....
Al_2O_3			1.31	.....			.....	.....	10.00	5.40	.....
NiO	0.08	Trace	.....	.....	.....	.....	.....	.....	.....	.....	.....
CoO	0.09	Trace	.....	.....	.....	.....	.....	.....	.....	.....	.....
CaO	8.74	6.62	5.26	.....	1.15	0.61	1.01	7.00	.....	2.63	.....
MgO	3.66	.....	None	.....	.....	.....	0.25	.....	.....	.....	.....
SiO_2	16.11	13.06	8.63	12.3	.....	.....	6.19	.....	9.00	2.25	4.6(*i*)
S	.....	.....	2.53	7.8	6.22	5.42	.....	.....	.....	.....	.....
SO_3	9.30	8.85	1.61	2.8	14.78	14.07	3.38	13.00	28.81	16.84	4.8
H_2O	1.28	3.76	11.20								6.9
CO_2				.....	.....	.....	.....	.....	.....	.....	.....
C	19.23	22.14		2.5	8.00	5.80	1.17	.....	.....	.....	.....

(*a*) Curtis, *Tr. A. I. M. E.*, 1873-74. II, 95. (*b*) Dewey, Bulletin No. 42, "United States National Museum," 1891, p. 53. (*c*) Balling, *Metallhüttenkunde*, 1885, p. 87. (*d*) Freudenberg, M., "Die auf der Bleihütte bei Ems zur Gewinnung des Flugstaubes getroffenen Einrichtungen," Ems, 1882, p. 19; Abstr. *Eng. Min. J.*, 1882, XXXIV, 14; Egleston, *Tr. A. I. M. E.*, 1882–83, XI, 379; Stetefeldt, "Comment on Freudenberg's Plates," *Eng. Min. J.*, 1883, XXXVI, 51. (*e*) Hering, *op. cit.*, p. 34. (*f*) French, *Eng. Min. J.*, 1880, XXIX, 49. (*g*) Kosmann, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1883, XXXI, 227. (*h*) As oxides. (*i*) Insoluble residue.

and Table 101 some more recent data from the works of the Globe Smelting and Refining Co., Denver, Colo., communicated by M. W. Iles.

The analyses do not take any cognizance of the presence of NH_3. This has been noticed by several metallurgists. Iles[1] observed it in icicles attached to the conduit pipes leading to the bag house. Canby,[2] Dwight,[3] and Semple[4]

[1] *Eng. Min. J.*, 1889, LXVIII, 576.
[2] *Loc. cit.*, p. 633.
[3] *Tr. A. I. M. E.*, 1912, XLIII, 746.
[4] *Eng. Min. J.*, 1913, XCV, 1208.

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TABLE 101.—ANALYSES OF FLUE-DUST FROM THE GLOBE SMELTERY, DENVER, COLO.

	Average sample of all dust collected in 10 years	Dust from downcomers of 11 blast furnaces	Dust from roof of blast furnace building	Dust from hood above slag tap	Fume from slag-pot while boiling	Fume from reverberatory settling furnace
SiO_2	26.20	3.65	.....	4.80	0.80	.....
Fe	13.40	4.60	.....	.....	.....	.....
Fe_2O_3	.....	.....	2.30	1.40	.....	.....
Mn	0.93	.....	.....	.....	.....	.....
CaO	5.30	2.80	.....	.....	.....	.....
BaO	4.60	.....	.....	.....	.....	.....
Sb	.....	.....	.....	.....	.....	2.20
SbO_3	.....	.....	.....	.....	1.80	.....
As	.....	.....	.....	.....	.....	2.30
As_2O_3	.....	.....	.....	0.64	4.80	.....
Zn	4.74	2.70	.....	.....	.....	26.70
ZnO	.....	.....	.....	32.50	22.20	.....
Cu	0.95	.....	.....	.....	.....	.....
Pb	25.60	47.50	27.10	.....	.....	31.00
PbO	.....	.....	.....	28.50	41.00	.....
$PbSO_4$	.....	.....	.....	27.10	26.20	.....
S	4.20	.....	.....	.....	.....	6.30
SO_3	.....	1.70	.....	.....	.....	.....
Ag, oz	33.40	16.70	25.40	.....	.....	4.60
Au, oz	0.14	0.06	0.10	.....	.....	Trace

found that when flue-dust was mixed with milk of lime previous to briquetting, NH_3 was set free. This presupposes that the NH_3, present in the combined state, is liberated by the $Ca(OH)_2$. Richards[1] suggests that AlN may be the source of NH_3. Hickman[2] seeks it in the volatile matter of the coke, which has not been completely expelled; the burning of the coke would be completed in the upper part of the blast furnace, and the NH_3 present expelled; the latter would combine with SO_3 to $(NH_4)_2SO_4$, and this would be collected, at least in part, in the flue-dust.

The flue-dust from blast furnaces has generally a dark color, which is caused by the admixture of finely divided fuel. It is black when charcoal forms even a small part of the fuel. The amount of flue-dust formed depends upon a variety of causes. Fine ores or fluxes are carried away easily by the current of gases; charcoal, being friable, makes dust; soft coke is broken up to some extent, and causes mechanical losses; and the manipulation of the furnace, affecting the descent of the charge, has a very great influence. Thus careful feeding and cutting out of wall accretions will reduce the formation of flue-dust to a great extent. Then, a high temperature in the smelting zone causes much volatilization; in the same way a high-pressure blast will cause much vapor to be

[1] *Tr. A. I. M. E.*, 1912, XLIII, 748.
[2] *Eng. Min. J.*, 1914, XCVII, 1260.

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carried out of the furnace, if the quick ascent of the gases be not checked by the form of the furnace (by having boshes). Extreme figures of the amount of flue-dust formed are 0.8 and 15 per cent. of the weight of the ore charged.[1] An average figure used to be 5 per cent.; this has fallen to 2 per cent. since blast roasted material has formed 70 to 80 per cent. of the charge.

The character and consequent value of flue-dust changes greatly according to the part of the flue in which it is collected. Near the blast furnace it will resemble very much the ore that was charged, further on it will grow richer in Pb and lower in Ag until all ore particles have been settled out, and only fume remains suspended in the gas current. This will be very rich in Pb and very poor in Ag. Thus filtered fume coming from blast furnaces producing lead bullion assaying from 300 to 500 oz. Ag per ton will assay only about 4 oz. This was shown very clearly in the F. L. Bartlett process (§73). It may be further illustrated by two instances, one of ample and one of insufficient condensation. The dust collected at a distance of 625 ft. from the blast furnace contained from 20 to 25 per cent. Pb and 50 oz. Ag per ton; after passing the rest of the way, a flue 5010 ft. long, it contained 65 per cent. Pb and 1 oz. Ag per ton. In the second instance, the dust behind the blast furnace assayed 35 per cent. Pb and 31 oz. Ag per ton; in the main dust chamber, 41 per cent. Pb and 26 oz. Ag per ton; at the foot of the stack, 52 per cent. Pb and 17 oz. Ag per ton.

The flue-dust from roasting and reverberatory smelting furnaces, blast roasting apparatus, and converters is light colored, as the processes are oxidizing; the dust and fume are therefore composed of oxidized materials, and contain practically no carbonaceous matter. The amount formed varies too much to permit making a general statement; numerical data have been given with the several furnaces.

197. Recovery of Flue-dust and Fume; General.[2]—The methods in operation at lead smelteries and refineries may be classed as dry, wet, and electric. Dry methods of collection are used almost exclusively in the United States and do excellent work. Wet methods were in operation at the Cheltenham Works of the St. Louis Smelting Works;[3] they have found much favor in European works. Electric condensation of fume by means of the Cottrell process is making much headway in lead plants and promises in time to replace the bag house which at present serves this purpose.

Whatever method may be used for collecting dust and fume, the gases issuing from the furnaces travel through flues and perhaps chambers in which they drop most of their dust, and then pass on to special condensation apparatus.

198. Dry Collection of Dust and Fume.—The steps necessary for satisfactory work are, cooling of gas and retardation of its velocity with settling of dust, to be followed, when necessary, by filtration for the collection of fume.

199. Cooling of Gas.—Cooling of gas is essential to reduce the temperature

[1] Hahn, *Min. Res. U. S.*, 1882, p. 344.

[2] Iles, *School Min. Quart.*, 1896, XVII, 977.
Hofman, "General Metallurgy," 1918, 831.

[3] Iles, *loc. cit.*, p. 105.

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to a point at which volatilized metal or metallic compound ceases to exist in the state of vapor. The single particles suspended in the gas current are brought together more closely, and unite into flaky masses which settle more or less readily. Cold surfaces attract particles; hot surfaces, specially when moist, repel them.[1]

Though the gases from a lead blast furnace run with a cool top require very little cooling, the fine dust and vapor coming from reverberatory smelting and refining furnaces are likely to be lost, if their temperatures are not reduced.

In a current of air Pb is volatile at above 500° C.; PbO at 800°; PbS sublimes in a current of N at 800°; ZnO requires a higher temperature than PbO. The saturation curve for As_2O_3,[2] of Elton,[3] is given in Fig. 416. It shows, *e.g.*, that at 833° F. (=445° C.) 1 cu. ft. under standard conditions will hold 0.325 lb. As_2O_3, and at 250° F. (=121° C.) only a trace. The gas current must therefore be cooled to 121° C. if all the As_2O_3 is to be condensed. The gas SO_2 cannot be condensed by ordinary cooling methods. The vapor SO_3 is present in gases uncombined with H_2O at 440° C.; between 350 and 440° about half the vapor is in combination with H_2O; at about 338° the SO_3 has been wholly changed into H_2SO_4, but is still in the state of vapor; between 200 and 338° the vapor is in part converted into a mist; at 200° it begins to be completely condensed, *i.e.*, H_2SO_4 of 62° Bé. (sp. gr. 1.75) begins to be vaporized at this temperature. In general it may be said that, if gases leave the stack at a temperature twice as high as that of the surrounding atmosphere, the cooling has been carried as far as it is possible without resorting to induced draft.

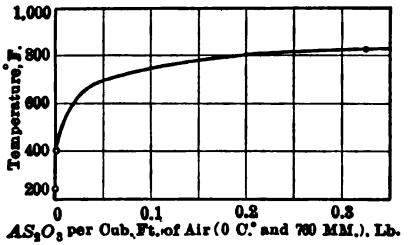

FIG. 416.—Vapor-tension curve of As_2O_3.

The common medium for cooling flues and chambers is atmospheric air; in some instances this has been supplemented by the use of water.

At present flues are built of sheet iron and brickwork. Formerly reinforced concrete, a coarse lattice of iron wire, or expanded metal imbedded in cement concrete,[4] was much in favor[5] on account of the efficient cooling by the thin walls, but has fallen into disuse, as the walls cracked and were attacked by sulphurous gases. In an iron pipe there is a loss of 0.02–0.08 B.t.u. per sq.

[1] Aitken, *Proc. Roy. Soc.*, Edinburgh, 1913, XXXII, 183.
Russell, *Phil. Trans. Roy. Soc.*, 1903, CCI, A, pp. 185, 551.

[2] Welsh, H. V.—Duschak, L. H., "Vapor Pressure of Arsenious Oxide," Bur. Mines, Techn. Paper 81, Washington, 1915.

[3] *Tr. A. I. M. E.*, 1913, XLVI, 697.

[4] Monier flue: Bauer, *Freiberg. Jahrb.*, 1894, 39; *Eng. Min. J.*, 1895, LIX, 342; *Min. Ind.*, 1895, IV, 478.

[5] Messiter, *Eng. News*, 1899, XLII, 356; *Min. Ind.*, 1900, IX, 455.
Edwards, *Tr. A. I. M. E.*, 1905, XXXV, 60, 965.
Welsh, *Eng. Min. J.*, 1904, LXXVIII, 348.
Hain, *Eng. News*, 1905, LIII, 279; *Min. Ind.*, 1905, XIV, 417.

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ft. per min. for 1° F. difference (=9.05 to 36.2 Cal. per sq. ft. per min. for 1° C.) between the outside of the pipe and the surrounding air;[1] Anderson[2] gives the rate of transmission, through sheet-steel plates about ⅛ in. thick, as 0.02 to 0.04 B.t.u. per sq. ft. per min. per 1° F. (=9.05 to 18.1 Cal. per sq. ft. per min. per 1° C.). The values, however, vary with the cleanliness of flue, convection of wind, exposure to sun, etc., and increase with the temperature of the pipe. In a brick flue with sides 13 in. and roof 9 in. thick there is in a 100-ft. flue a loss of 0.154° C. for every degree difference between gas inside and air outside.[3]

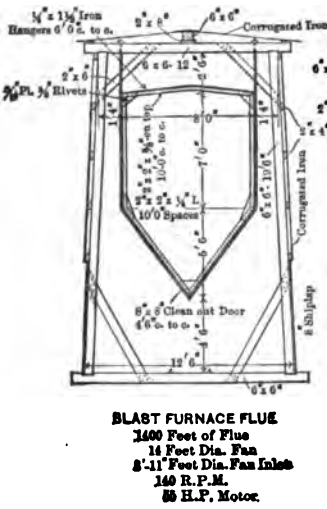
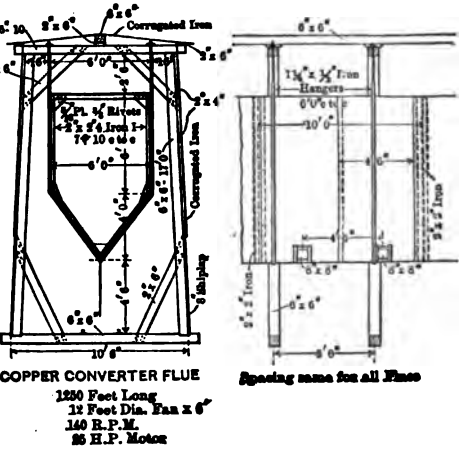


FIG. 417.—Suspended sheet-iron blast furnace flue, Omaha, Neb.



FIGS. 418 and 419.—Suspended sheet-iron converter flue, Omaha, Neb.

In Fig. 417[4] is shown the suspended sheet-iron blast furnace flue of Omaha, Neb., which is 1400 ft. long. The gases from the blast furnaces enter a brick flue before they pass into the sheet-iron flue which is connected with a fan delivering the cooled gas to the bag house.

Figs. 418 and 419 represent the sheet-iron converter flue of Omaha, Neb.,[5] which is 1250 ft. long. The gases are drawn through the flue by a fan at the rate of 80,000 cu. ft. per min. They enter the flue with a temperature of 371° and leave it at 121° C.

The supported balloon-shaped flue of Tooele, Utah, is shown in Figs. 420–424. It is made of plate iron ⅛ in. thick, and carries the gases from the blast furnaces to the bag house. A volume of 88,000 cu. ft. per min. enters the flue at 102° C., travels at the rate of 18 ft. per sec., and leaves at 84° C.

[1] Norton, C. L., private communication, January, 1918.
[2] *Tr. A. I. M. E.*, 1916, XLIX, 575.
[3] Lee, *Bull. Tech. Eng. Soc. Colo. School Min.*, 1909, IV, 197; *Min. Ind.*, 1909, XVIII, 497.
[4] Eilers, *Tr. A. I. M. E.*, 1912, XLIV, 708.
[5] Eilers, *loc. cit.*

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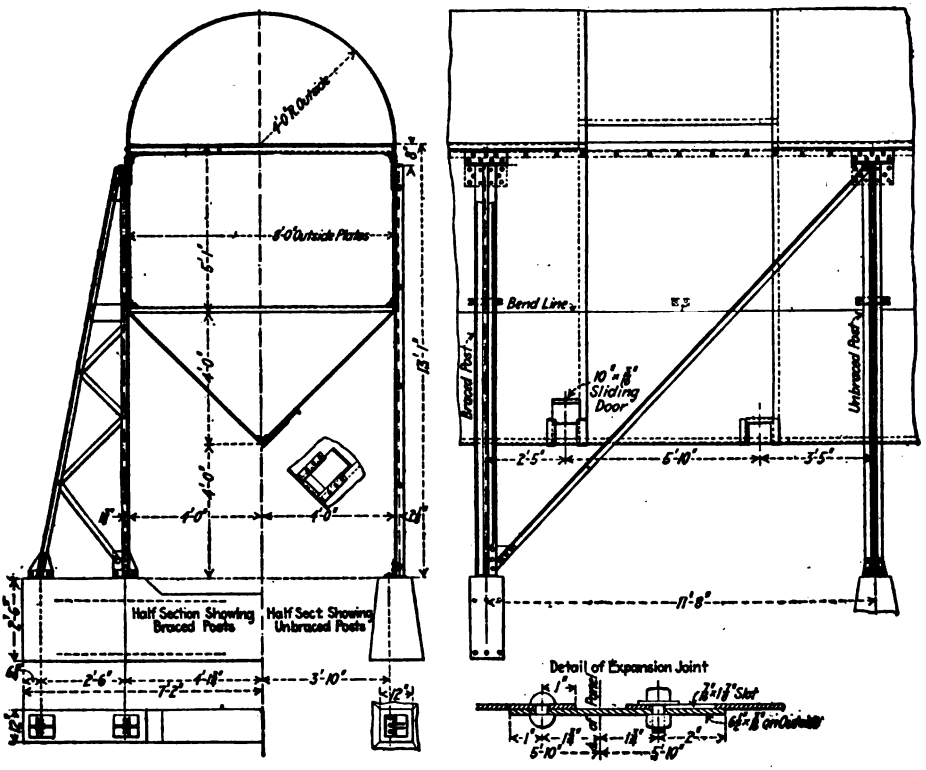
Iron flues are usually coated with graphite in order to protect them against corrosion.

Brick flues usually have one of the two following forms:[1] One is shown in Fig. 425. This represents the flue for roaster-gases at Durango, Colo., which has vertical sides, 13 in. thick, and a low arched roof, 9 in. thick. The footing of the walls is 15 to 24 in. deep and 18 to 24 in. wide; the buckstays and tie-rods are of soft steel. The other, having in cross-section the form of a catenary curve, is given in Fig. 426. It is built of 9-in. brick, strengthened at intervals



FIGS. 420 to 424.—Supported sheet-iron balloon flue, Tooele, Utah.

of from 6 to 10 ft. by ribs of reinforced concrete; expansion joints, 2 in. wide, are provided every 100 ft., and are covered with one thickness of brick laid dry. The most economic section is obtained by making the height 75 per cent. of the base, and the cross-sectional area 55 per cent. of the square of the base.

In the brick flue at Murray, Utah,[2] the blast furnace gases entering with a temperature of 48° C. showed a drop of 19° C. in 865 ft. when the outside air had a temperature of 18° C. The length of the flue is 2000 ft.

In some European plants air-cooling has been assisted by external water-

[1] Lee, *Bull. Tech. Eng. Soc. Colo. School Min.*, 1909, IV, 197; *Min. Ind.*, 1909, XVIII, 497.
[2] Eilers, *loc. cit.*

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cooling. At Freiberg, Saxony,[1] oblong flues of sheet lead used to be cooled with water admitted onto the roof which trickled down over the sides. At Tarnowitz, Silesia,[2] bundles of pipes with circulating water used to be suspended through the arch of a brick flue; adhering fume was hosed off at intervals. At other places cooled solutions were circulated in coils of iron pipe placed at certain intervals in the flue.[3]

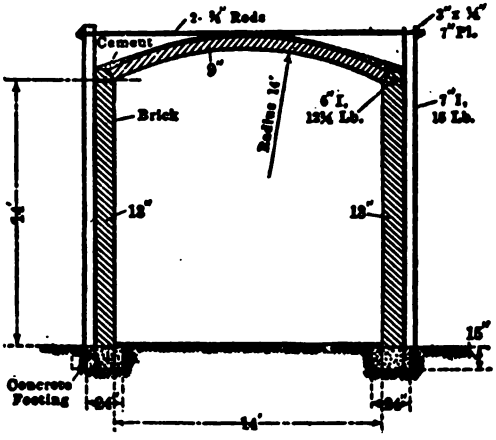

FIG. 425.—Brick dust-flue, straight side-walls.

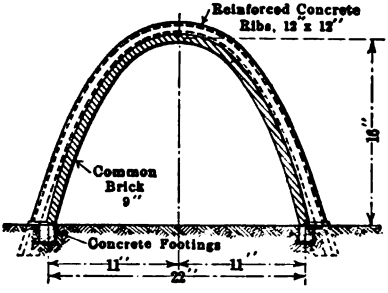

FIG. 426.—Brick dust-flue, catenary wall.

200. Retardation of Gas Current and Settling of Dust.—The gases in the downcomer of the blast furnace have a velocity of 20 to 30 ft. per sec. Recent investigations by Shelby,[4] Moore and Wright,[5] Lee,[6] Goodale and Klepinger[7] in connection with copper blast furnaces have shown that dust is settled satisfactorily, if in a clear open flue or chamber the velocity is reduced to 2½ ft. per sec. and the gases remain 50 sec. in it, or to 5 ft. per sec. in a flue or chamber hung with wires. With lead plants which collect unsettled fume by filtration or electric precipitation the velocity need not be reduced to below 4 or 5 ft. per sec. provided the gases remain 70 sec. in the flue or chamber. The chimney velocity should not be greater than 20 or 25 ft. per sec.

Retardation of gases is usually accomplished by change of direction of flue; by suspension in flue of plates at right angles to gas-current; by enlargement of flue to chamber; by alternate enlargement and contraction of flue; by suspension of plates in flue parallel with gas current; by suspension of wires in flue. The relative efficiency of these dust-arresting devices has been determined experimentally by Goodale and Klepinger;[8] their results are given below in §206.

[1] Hagen, *Freiberg. Jahrb.*, 1879, p. 171.
[2] Saeger, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1893, XLV, 280.
[3] Schlösser-Ernst, *Berg. Hüttenm. Z.*, 1885, XLIV, 464; 1887, XLVI, 134.
[4] *Eng. Min. J.*, 1908, LXXXV, 205.
[5] *Op. cit.*, 1910, LXXXIX, 449; XC, 104, 111, 112.
[6] *Op. cit.*, 1910, XC, 504.
[7] *Tr. A. I. M. E.*, 1913, XLVI, 567.
[8] *Tr. A. I. M. E.*, 1913, XLVI, 589.

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201. Change of Direction of Flue.—In the older European lead works it was customary to build long brick flues; these cooled the gases and retarded the velocity on account of the friction between stationary walls and traveling gas. Thus, at Freiberg, Saxony, there are in operation flues 5 miles long. In order to decrease the length, zig-zag flues, as shown diagrammatically in Fig. 427, have been built. It was found that some dust accumulated at every change of direction of gas current; they were therefore an improvement on the straight long flue as regards collection of dust, and at the same time caused a saving in building material; but the additional amount of dust collected is small. As long as such zig-zag flues are built on pillars or are suspended so that air can play around them, they are effective. Thus, the customary balloon-flue (Figs. 417–424), usually built in a straight line, can be erected to cover a smaller floor-space.

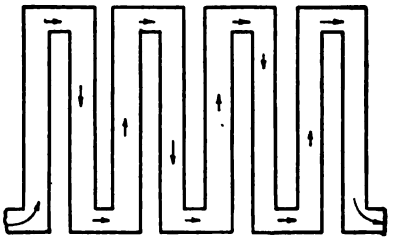

FIG. 427.—Zig-zag flue, plan.

202. Suspension in Flue of Plates at Right Angle to Gas Current.—Plates can be introduced into a flue in a manner to force the gas current to travel in a sinuous path either in a horizontal or a vertical plane. Both arrangements have an effect similar to that of a zig-zag flue excepting that the temperature is not reduced as effectively. Gases traveling along a serpentine path in a horizontal plane will not drop dust as readily as when their course is deflected slightly up and down, as here the dust dropped on the downward travel can be collected in a pocket in which there is no movement of air. Many European plants have adopted this construction. See curves 35 and 36 in Fig. 437.

203. Enlargement of Flue to Chamber.—Increase of size of a flue to a chamber effectively decreases the velocity of the gas-current and thereby favors settling of dust. Upon entering the chamber, the velocity is slackened gradually to the point where the draft near the exit begins to show its effect; then there is a gradual increase in velocity. The chamber, however, should not be too large,[1] as the gas current does not expand to fill the entire chamber; it travels only through part of it, and then only part of the enlarged flue or chamber is really utilized in the settling of dust. It has been suggested that the approach to the chamber should be tapered, as the dispersion of the gas-current in the chamber would be increased and thus a larger part utilized for settling.[2] See curves 32 and 34 in Fig. 437.

204. Alternate Enlargement and Contraction of Flue.—If, as shown in (§203), a single great enlargement of flue is not likely to utilize to the full extent the cross-sectional area of a chamber, repetition of the process with small enlargements ought to be more satisfactory. So far this method has not found much application. See curve 37 in Fig. 437.

[1] Messiter, *Min. Sc. Press*, 1908, XCVII, 26.

[2] Wedge dust chamber, *Eng. Min. J.*, 1916, CI, 646.

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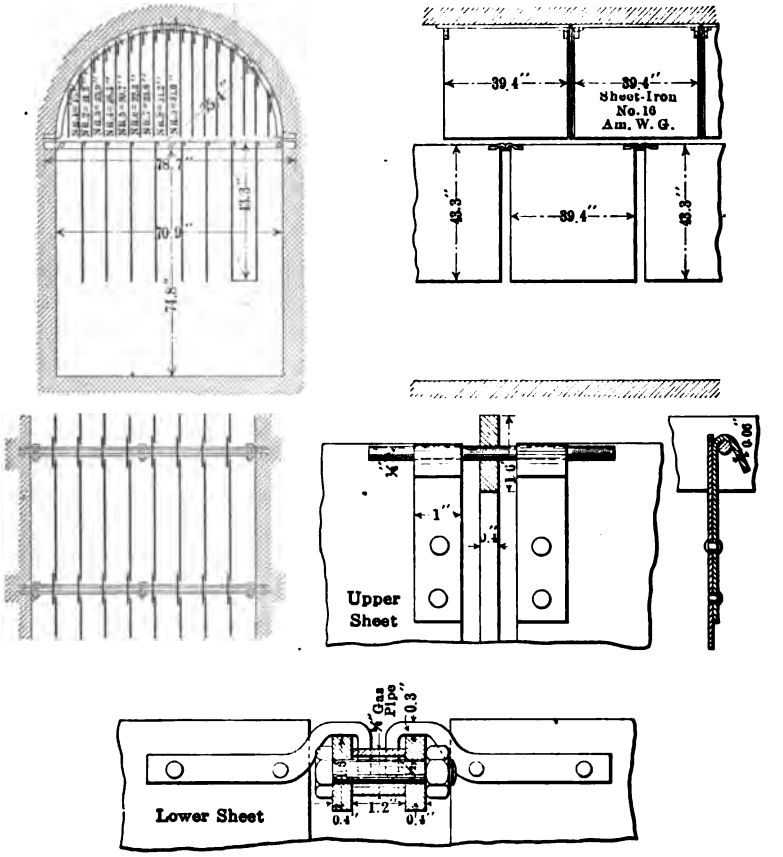
205. Suspension of Plates in Flue Parallel with Gas Current.—In the apparatus referred to there has been passed over the consideration of the friction between stationary walls of flue and moving gas current, which is the cause of the reduction of velocity. It is Freudenberg's merit[1] to have discovered, that an increase of surface is an effective means of settling dust, and that the amount of settled dust is in direct proportion to the area of surface with which



FIGS. 428 to 433.—Flue with Freudenberg plates, Freiberg, Saxony.

the gas comes in contact. The cooler the surface, the greater will be its attraction for suspended particles.[2] In order to increase the surface area, Freudenberg suspended at the lead works of Ems, Prussia, thin sheet-iron plates[3] parallel with the gas-current, and to prevent settled dust from being carried off, he

[1] Freudenberg, M., "Die auf der Bleihütte bei Ems zur Gewinnung des Flugstaubes getroffenen Einrichtungen," Ems, 1882; Abstr., *Eng. Min. J.*, 1882, XXXIV, 1.
Egleston, *Tr. A. I. M. E.*, 1882–83, XI, 379.
Stetefeldt, "Comment on Freudenberg's Plates," *Eng. Min. J.*, 1883, XXXVI, 51.

[2] Aitken–Russell, p. 431.

[3] Illustrated in reference.

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placed vertical partitions across the bottom which reached nearly to the hanging plates. The plates are self-cleaning, that is, the dust collecting on the plates falls off when it has grown to a thickness ranging from 1⅛ to 1¼ in. Figs. 428–433 show the Freudenberg plates of Freiberg, Saxony.[1] All plates are 3 ft. 3⅜ in. wide and 1/32 in. thick. The plates in the arched portion of the flue (from 8 to 16 in number) are from 1 ft. 3 in. to 2 ft. 10¼ in. long. They are suspended at intervals of 4¾ in. by hooks from pins passing through band-iron which, following the sweep of the arch, is let in at the sides and fastened to the roof. There are half as many plates in the lower part of the flue; they are 3 ft. 7¼ in. long and about 8 in. apart. They are supported by lugs, riveted to the upper ends on cross-bars let into the sides. Thus when enough dust has collected on the bottom of the flue to make a clean-up, the lower plates are pushed to one side, and then do not interfere with the work. A comparison of 3 years' work both with and without Freudenberg plates shows that the amount of flue-dust collected both in the flues leading to the chamber containing the plates and in the chamber itself is twice as much with the plates as without them, and that the dust in the flue leading from the chamber to the stack is much less. A comparison of the assay values shows that the dust settled out in the flue leading to the chamber was richer in Pb and As than when no plates were used, and that in the chamber proper and the flue leading from it to the chimney more As was collected and less Pb. These results speak very well for the plates. The disadvantages are that the plates are quickly corroded by acid moisture if the temperature of the gases sinks below 50° C., that they diminish the natural draft so that forced draft becomes necessary, and that they are expensive. Curve 33 of Fig. 437 shows how effective the plates are when compared with other arresting devices.

An older system of increasing the flue-surface is represented by the Cowles flues.[2] Here the gas from the main flue passes on either side into a number of smaller branches placed at angle of about 45°; these unite into sub-conduits which end in the single main.

206. Suspension of Wires in Flue.—The success of Freudenberg prompted Roesing[3] to suspend wires in the dust-flue of Tarnowitz, Silesia, which did most satisfactory work. The wires are of No. 10 gauge and 10 ft. long. The manner of suspending them is shown in Figs. 434–436. These represent a wire screen, *e*, of about 1-in. mesh, riveted to I-beams, *d*, running along the top of a flue which is 16 ft. 6 in. high. From the screen the wires are suspended. The gases travel upward in one chamber and downward in the next, that is, parallel with the wires. By a shaking arrangement the dust can be more easily removed from the wire than from the plates. At Tarnowitz, Silesia, during 300 working days in 1888 there passed off into the open air without the wire system 121,000 lb. of solid matter for every furnace; with the wire system the weight of the solid matter was reduced to 90,200 lb. without diminishing the draft of the chimney.

[1] Bauer, *Freiberg. Jahrb.*, 1894, 39; *Eng. Min. J.*, 1895, LIX, 342; *Min. Ind.*, 1895, IV, 476.
[2] Iles, *School Min. Quart.*, 1895–96, XVII, 115.
[3] Saeger, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1895, XLI, 267.

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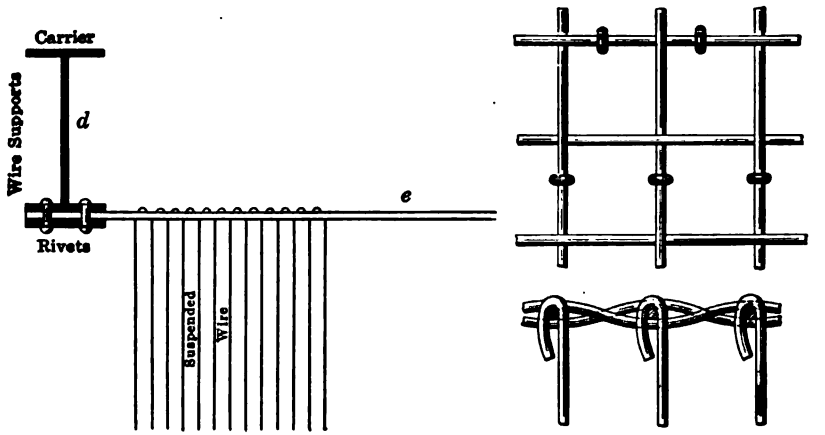
In 1900[1] the Anaconda Copper Mining Co. completed at its plant of Great Falls, Mont., the new flue-system of which the Roesing wire-system forms an integral part. The paper has to be consulted fully to study this admirable system of collecting flue-dust. The part that is of main interest here is the dust chamber 367 by 177 ft. and 21 ft. high from the tops of the hoppers, which form the floor, to the roof. The chamber is hung for a distance of 317 ft. with 1,200,000 steel wires spaced 2.3 in. The gases travel at right angles to the suspended wires. The wires, weighing each about 1 lb., are hung in two groups with a clear space, 47 ft. long, intervening, for the purpose of collecting dust in the entrance, and fume in the exit division. At the inlet end there is left a clear space for the even distribution of the gases; in



FIGS. 434 to 436.—Roesing wire-system, Tarnowitz, Silesia.

the first 51 ft. of the chamber the wires are 0.162 in. in diameter and 16 ft. long; the rest of the wired space has wires 0.135 in. in diameter and 20 ft. long. At the upper end of the entrance division there are 22 pipes for the admission of air to cool the gases to the desired temperature. The wires are suspended from a netting of steel wire, with 1⅝-in. openings, bolted to the I-beams of the roof. The suspended wires are shaken for 30 min. at intervals of 60 to 90 days by means of angle-frames, suspended by hangers, through connecting rods extending through the flue-walls and attached to a bell-crank lever. The efficiency of the wire-system is shown in curves 38 and 39 of Fig. 437.

207. Efficiency of Dust-arresting Devices.—The tests of Goodale and Klepinger were carried on in an experimental flue, 4 ft. wide by 4 ft. 6 in. high and 304 ft. long, having a furnace-gas inlet at one end and an exhaust-fan at the other. The velocity of the gases ranged from 4.4 to 9.4 ft. per sec., and the temperature was 400° F. (204° C.) The results obtained in the first 300 ft. of the chamber are shown in Fig. 437. The ordinate at the left gives the pounds of dust and fume collected per 1,000,000 cu. ft. gas at 400° F.

[1] Goodale-Klepinger, *Tr. A. I. M. E.*, 1913, XLVI, 567.

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(204° C.), the one at the right the percentage of the total dust recovered, the total of 100 per cent. being arrived at by filtering a measured volume of gas through mineral wool. The abscissa gives the length of the flue under consideration.

Tests 32 and 34 show that in the open flue, without any obstruction whatever and with velocities of gas of 9.4 and 8.9 ft. per sec., there were collected in 300 ft. of chamber between 30 and 40 per cent. of the dust.

Test 33 indicates that over 40 per cent. of the dust was recovered, when 11 rows of longitudinal surface plates (25 by 35 in. and 0.5 in. thick), *i.e.*,

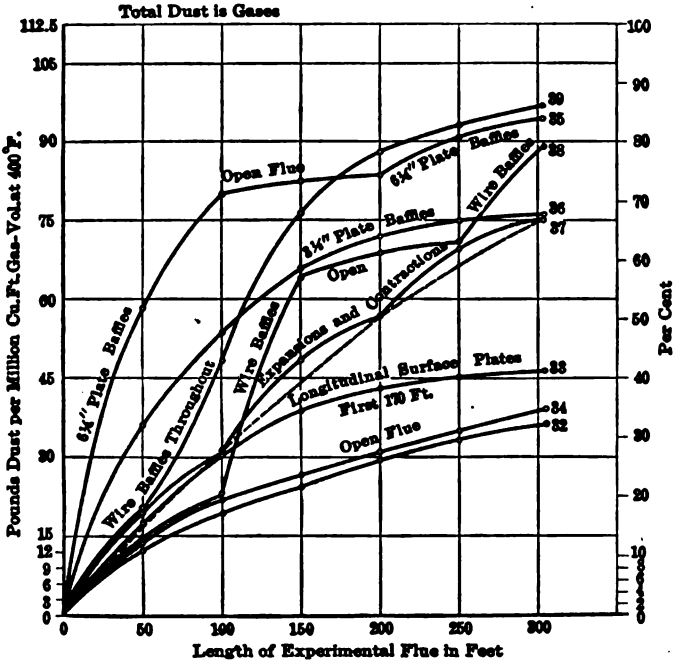

FIG. 437.—Relative efficiency of dust-arresting devices, Great Falls, Mont.

Freudenberg plates, were hung 4 in. apart in the flue for a distance of 170 ft.; cross-plates, 8 in. high, were placed 10 ft. apart on the floor. The velocity of the gas was 9.3 ft. per sec.

Tests 35 and 36 exhibit the effects of 6¼-and 3⅛-in. plate baffles placed staggeringly across the flue so that they present their surfaces to the gas-current; plates, 8 in. high, were placed every 10 ft. on the floor to arrest settled dust. In test 35, the baffles, 6.25 in. wide, were suspended 7 in. apart, with open spaces 5¾ in. between baffles and 2⅞ in. between baffles and walls, in two divisions, 92 ft. long, separated by a clear space of 100 ft. In test 36, baffles, 3⅛ in. wide, were used throughout the flue with the exception of about 8 ft. at the end. The 6.25-in. baffles effected a saving of over 80 per cent. of the dust with a gas-velocity of 4.4 ft. per sec.; the 3⅛-in. baffles one of 70 per cent. with a velocity of 5.7 ft. per sec.

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Test 37 displays the effects of expansions and contractions of the flue. The flue, 18 sq. ft. in cross-sectional area, was partly closed at 100 and 104 ft. from the ends by two partitions, each having a central pipe 18.5 in. in diameter or a cross-sectional area of 1.8 sq. ft. This simple contrivance caused a saving of over 70 per cent. of the dust with a gas-velocity of 6.9 ft. per sec.

Tests 38 and 39 bring out clearly the great saving of dust accomplished by wire-baffles or the Roesing system. Copper wires, No. 7 B. & S. gauge, were suspended from poultry netting at 2- and 2.5-in. centers to extend from roof to floor. In test 38, two 50-ft. sections were hung with wires; it presents a saving of close on to 80 per cent. of the dust with a gas-velocity of 7.9 ft. per sec., and demonstrates in a striking manner the difference between a wired and a clear flue. In test 39, the flue was hung throughout with wires with the exception of 9 ft. at the end; it shows the greatest saving, about 85 per cent., with a gas-velocity of 7.3 ft. per sec.

As a result of these tests, the new dust chamber of Great Falls has been provided with wire-baffles as outlined.

208. Filtration of Fume, General.[1]—The fumes of lead furnaces which have not been collected with the dust are recovered at present mainly by passing them through suspended cotton or woolen bags. These allow the gases to pass freely through the meshes, but retain the fume and the fine dust which has remained in suspension owing to imperfect settling.

Cloths stretched up and down in zig-zag across a dust-chamber (Brown-DeCamp fume collector) were once used as a filter at Omaha, Neb.[2] The filtering was satisfactory, but the difficulty and expense of making repairs in the cloth were too great to warrant continuing work with this device.

A bag house[3] will be used for filtering if the values recovered are sufficiently large to pay for the installation and the cost of operating, both of which are high,[4] or if it is necessary to prevent metallic compounds from passing off into the atmosphere which would do damage to the country in the neighborhood; it may serve also to prevent losses during abnormal workings of the furnaces. Unfortunately filtering recovers all the fumes, hence also As_2O_3, a by-product which it is often difficult to market.

It has been shown that in lead blast furnaces run with cold tops the loss in metal is caused mainly by particles of dust, as the furnaces make very little fume. As long as there has been made adequate provision for settling dust, a bag house will not be needed.

Eilers,[5] discussing the bag house of the smeltery at Murray, Utah, which smelts silver-lead ores with charges containing 10 to 12 per cent. Pb, and that of the smeltery at Omaha, Neb., which treats in the blast furnace charges running

[1] Iles, *loc. cit.*
Ebaugh, *J. Ind. Eng. Chem.*, 1909, 1, 686; *Eng. Min. J.*, 1909, LXXXVIII, 1020.
[2] *Eng. Min. J.*, 1895, LX, 372.
[3] Iles, M. W.; "Lead Smelting," Wiley, New York, 1902, pp. 186–211.
[4] A bag house with 4000 bags costs about $150,000.
Editor, *Eng. Min. J.*, 1909, LXXXVIII, 1021.
[5] *Tr. A. I. M. E.*, 1912, XLIV, 708.

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high in Pb and Ag, has shown that the bag house of Murray does not pay expenses, whereas that at Omaha furnishes adequate returns. The experience in other localities differs in part from that of Eilers in that, *e.g.*, at Herculaneum, Mo., a bag house is being erected to treat the gases from the blast furnaces which treat charges running high in Pb and practically free from Ag.

In roasting, blast roasting, and converting there is formed a considerable amount of fume which carries too much value to allow it to pass into the open without filtering (or electric precipitation). The same is the case with refineries which subject lead bullion to an oxidizing fusion. Gases from roasters and converters are rich in SO_2, and contain more or less SO_3 which corrodes animal and vegetable fibres; they have to be subjected to a special treatment (Sprague process §212) before they can be filtered. Bags are also corroded by the presence of Se in the gases,[1] which occurs as SeO_2, and coming in contact with SO_2 converts this into corrosive SO_3 as shown by $SeO_2+2SO_2=Se+2SO_3$. The collected fume always shows an acid reaction.

Filtering gases from the lead blast furnace was first tried in the early eighties at the Grant Smeltery of Leadville, Colo.,[2] but given up, as the filtered fume, though running high Pb, assayed only 6 to 10 oz. Ag per ton. The first prominent bag house plant was installed at the Globe Works of Denver, Colo., in 1890,[3] and has been in operation ever since. Other smelteries have followed suit, less on account of the values recovered than to conform to the laws of a number of states, and to be protected against possible suits for damages.

In order to ascertain approximately the losses due to fume, and with it to fine dust, a small bag house, such as the one constructed by C. C. Hoke at Mapimi, Mexico,[4] is valuable. More accurate results are obtained by the plant devised by J. M. Samuel[5] at Douglas, Ariz.

The bag house of Murray, Utah; the solenoid bag-shaking device; and the bag house of Midvale, Utah, may serve as samples of modern practice. The bag houses of Denver[6] and Pueblo, Colo.; East Helena, Mont.; and Selby, Cal.[7] resemble that of Murray; the one of Tooele, Utah,[8] that of Midvale; other filters, such as those of the Halberge Iron Works,[9] of Dupue, Ill.,[10] of Simon-

[1] Alexander, *Tr. A. I. M. E.*, 1914, XLIX, 565.

[2] Emmons, "Geology and Mining Industry of Leadville," Monogr. XII, U. S. Geol. Surv., Washington, 1886, pp. 673, 717.

[3] Ingalls, *Eng. Min. J.*, 1905, LXXX, 5.
Alexander, *Tr. A. I. M. E.*, 1914, XLIX, 561, drwgs.

[4] *Eng. Min. J.*, 1910, LXXXIX, 857.

[5] *Tr. A. I. M. E.*, 1916, LV, 751.

[6] Ingalls, *Eng. Min. J.*, 1905, LXXX, 5.
Alexander, *Tr. A. I. M. E.*, 1914, XLIX, 561.

[7] Bennett, *Eng. Min. J.*, 1908, LXXXVI, 115.
Hofman, "General Metallurgy," 1918, 854.

[8] Thomson-Licka, *Tr. A. I. M. E.*, 1913, XLVI, 942.

[9] Müller, *Stahl u. Eisen*, 1911, XXXI, 229; *Min. Ind.*, 1911, XX, 485.
Gouvy, *Rev. Un. Min.*, 1912, XL, 54.
Hofman, "General Metallurgy," 1918, 856.

[10] Brooks-Duncan, *Bull. A. I. M. E.*, November, 1917.

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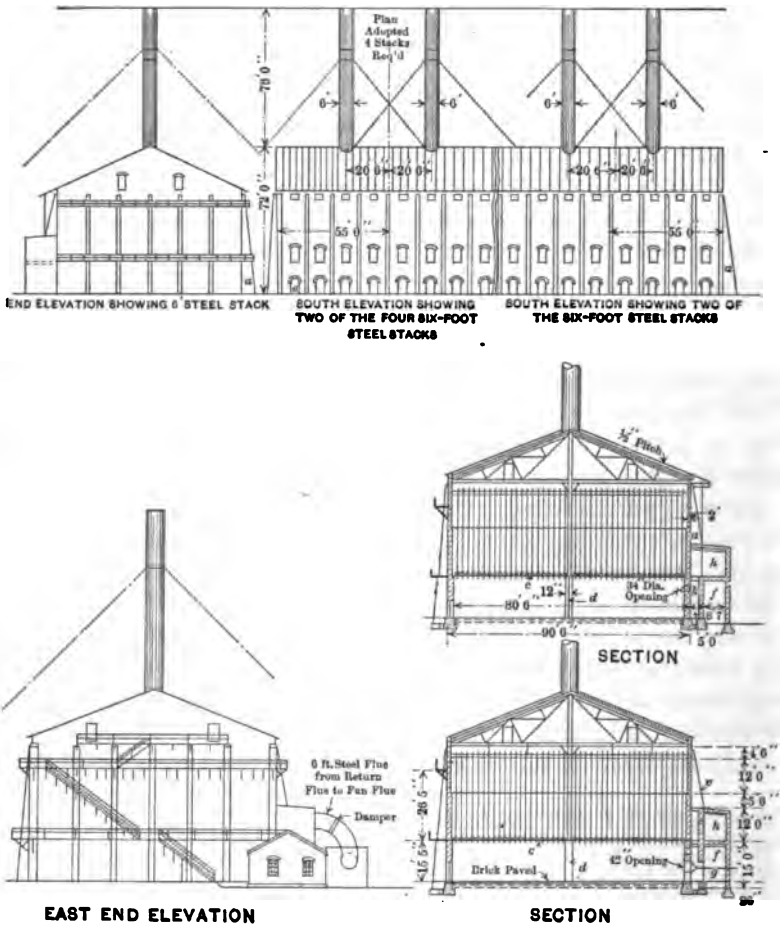
Buchler-Banmann,[1] of Riggs,[2] can only be noted, as is the case with the camels-hair filter of Cameron.[3]

209. Bag Filtration at Murray, Utah.[4]—This bag house, shown in Figs. 438–453, was erected in 1906 at a cost of $127,149.29, and went into operation in 1907. The elevations and sections, Figs. 438–442, give the leading features. The bag house, 216 ft. 6 in. by 90 ft. 6 in. and 51 ft. 6 in. high to roof trusses,


Plan Adopted 4 Stacks Req'd
END ELEVATION SHOWING 6 STEEL STACK
SOUTH ELEVATION SHOWING TWO OF THE FOUR SIX-FOOT STEEL STACKS
SOUTH ELEVATION SHOWING TWO OF THE SIX-FOOT STEEL STACKS
SECTION
6 ft. Steel Flue from Return Flue to Fan Flue
Damper
Brick Paved
42" Opening
EAST END ELEVATION
SECTION

FIGS. 438 to 442.—Bag filtration, Murray, Utah.

is built of brick; the outside walls are 21 in. thick for a height of 16 ft., are reduced to 17 in. for 18 ft., and to 13 in. for the remaining 17 ft. 6 in. Buttresses, 15 to a side and 5 to an end, are built into the walls to strengthen them; they are 24 in. thick, project 4 ft. at the floor-line and taper to 1 ft. at the line of the

[1] Ingalls, *Eng. Min. J.*, 1918, CV, 97.
[2] U. S. Patent, No. 1095076, May 5, 1914.
[3] *Met. Chem. Eng.*, 1917, XVI, 284; *Eng. Min. J.*, 1917, CIII, 1072.
[4] Eilers, *Tr. A. I. M. E.*, 1912, XLIV, 708.

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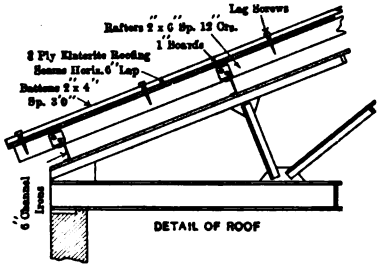
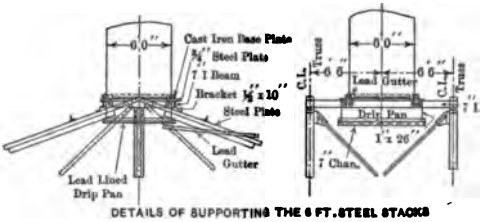
roof-trees. The details of construction of the roof are shown in Fig. 443. The bag house has four steel stacks, Figs. 438–442, 6 ft. in diameter and 78 ft. high, one for each compartment to draw off the gases. The moisture that condenses inside and trickles down is collected, Figs. 444 and 445, in a lead-lined pan; this is hung at a sufficient distance beneath the inlet so as not to interfere with the draft; from it a lead gutter runs out on to the roof.

The bag house is divided into four bays by three brick partition walls, *b*, Fig. 446, of the same construction as the outside walls, which extend from the floor-line to the apex of the roof. Below the thimble-floor, *c*, Figs. 441, 442, and 446, each bay is divided into four chambers or cellars by 13-in. brick partitions, *d*, Figs. 441, 442, and 446. A chamber, 87 by 12 ft., has four 8-ft. 12-in. doors, *e*, Fig. 439, for the removal of dust.

The distribution flue, *f*, Figs. 441 and 442, which is 16 by 16 ft., is built of brick and covered with a jack-arch roof. It runs along the entire length of the building, and is connected with the 16 chambers by means of 42-in. hinged dampers, *g*, Fig. 441, which are made of ¼-in. steel, stiffened by angle-irons and operated through 2-in. gas-pipe stems.



DETAIL OF ROOF



DETAILS OF SUPPORTING THE 6 FT. STEEL STACKS UPON THE STEEL BUILDING TRUSSES

FIGS. 443 to 445.—Bag filtration, Murray, Utah.

On top of the distributing flue is placed the return-flue, *h*, Figs. 441 and 442, which is 11 by 16 ft. and is connected with the 16 chambers by 16 ports, *k*, Fig. 442, provided with 34-in. hinged dampers; these open into the chambers, and are opened and closed by chains on the thimble-floor. The return-flue, *h*, Fig. 440, is connected with the fan flue from the blast furnaces, *i*, Figs. 440, 452, and 453, by a sheet-iron down-take provided with a damper in order that the gases from any one of the chambers, cut out by means of valve *g*, Fig. 441, to burn accumulated dust, may pass by port *k*, Fig. 442, into the fan and be distributed with the blast furnace gases in the other chambers.

In burning the dust in a chamber, the door, *e*, Fig. 439, is left open to furnish the necessary air.

The thimble-floor, *c*, Figs. 441, 442, and 446, made of No. 10 steel plate is 15 ft. 15 in. above the chamber floor. It is divided, as already shown, into four compartments, each of which contains 1008 steel thimbles, 17 in. inside diameter and 10 in. high. The thimbles are made of No. 14 steel plate and are riveted to the steel floor. The filter bags are slipped over them and wired.

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There are provided 4032 bags, Figs. 441, 442, 447–449, in the four compartments; they are 18 in. in diameter and 30 ft. long. They furnish a filtering surface of 570,012 sq. ft. which filters 165,000 cu. ft. gas per min.; this cor-

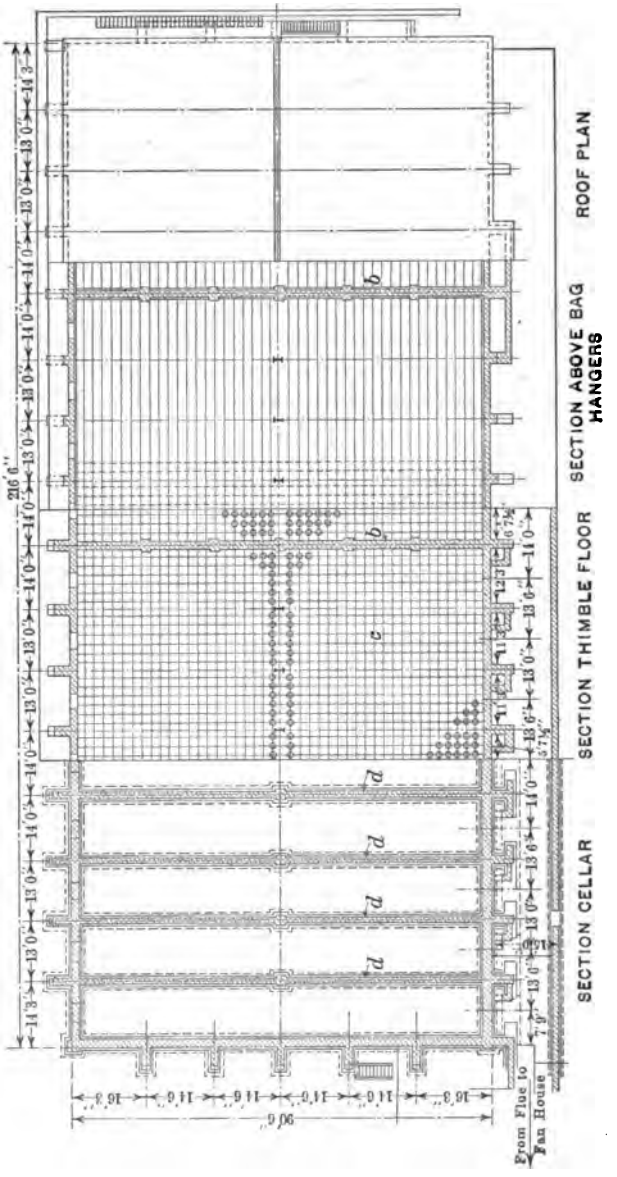

Fig. 446.—Bag filtration, Murray, Utah.

responds to 3.45 sq. ft. surface per cu. ft. gas per min. When one compartment is cut out for burning dust and fume in the chamber, there remain available 2.59 sq. ft. filtering surface per cu. ft. gas per min. The basis for calculating the number of bags needed was 2.5 sq. ft. per cu. ft. gas per

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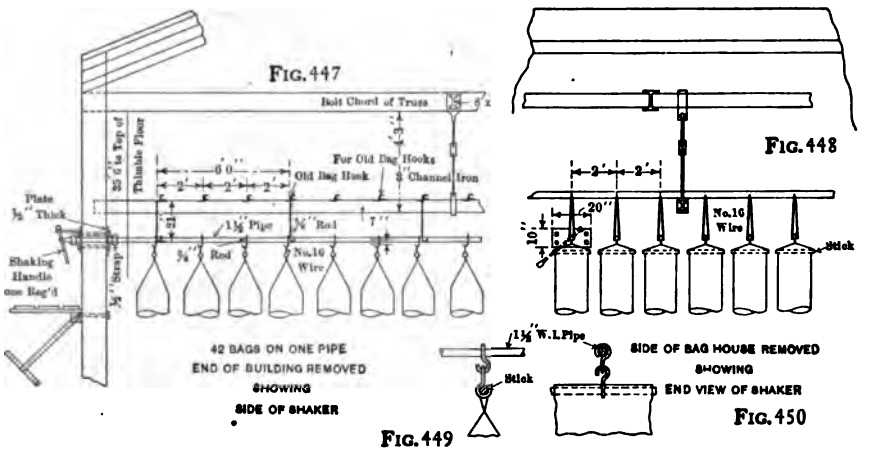
min. This is too little; at present 3 sq. ft. is the usual figure. The first bags used in 1907 were of cotton; they cost $2.136 apiece and showed an average life of 17 months 11 days. In 1909 woolen bags costing $4.7185 a piece were gradually substituted for the cotton; in 1913 the woolen bags were still in good condition.

The hanging and shaking of bags has undergone considerable change. At first, the bags were wired at the top and suspended by the wire from hooks fastened to 3-in. channel irons (Fig. 447, "for old Bag Hooks") and shaken by hand. For this purpose a compartment has to be freed from gas (aired out) before men could enter. In 1910 each row of 42 bags, Fig. 447, was suspended from a 1½-in. pipe hung by a ⅝-in. U-shaped rod from the old bag hooks. The pipe, extending outside the building, is there connected with a lever by means of which it secures a horizontal motion of about 5 in. and comes to a sudden stop by striking an iron plate, Fig. 448, on the inside of the brick wall.



FIGS. 447 to 450.—Bag filtration, Murray, Utah.

A vertical motion of about ½ in. amplitude is imparted to a bag while it moves in a horizontal direction. The bags are shaken, a compartment at a time, every 24 hr. for 3 to 4 min. Before shaking, all openings of a compartment are closed in order to prevent the temperature from falling too low. The shaking of all the bags takes a man about 3 hr. Dust accumulates in the folds of a wired bag and rots the cloth. This has been overcome by folding over a bag for 3 in. and sewing it to form a hem through which is passed a stick, Figs. 448–450; the stick is hung by an S-shaped hook. The bag is stretched by this means, and the wire does not come into contact with the cloth.

The fan house, Figs. 451–453, contains a Buffalo Forge exhaust fan with impeller 6 by 15 ft. It has a capacity of 225,000 cu. ft. gas running at a speed of 150 r.p.m., or 1500 cu. ft. per revolution, at a pressure of 1 in. water. The drawing shows a 75-h.p. induction motor. At first, a 100-h.p. motor drove the fan at a speed of 93.2 r.p.m., which took the gases from eight blast

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furnaces; later a 50-h.p. motor driving the fan at 80 r.p.m. took the gases from six blast furnaces. The blast furnace gas arrives in flue, *i*, and is delivered into distributing flue, *f*, at a pressure of ½ in. water; just before shaking the bags the pressure rises to ¾ in. The gas leaves a blast furnace through a sheet-iron goose-neck. The gases from eight blast furnaces pass through 1057 ft. of brick and 70 ft. of sheet-iron flue before they enter the fan house. Their average temperature is 38° C. and the SO_3 content <0.003 per cent.

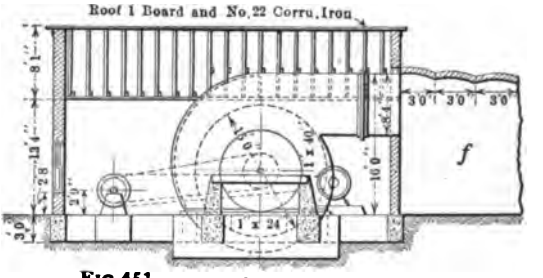

FIG. 451

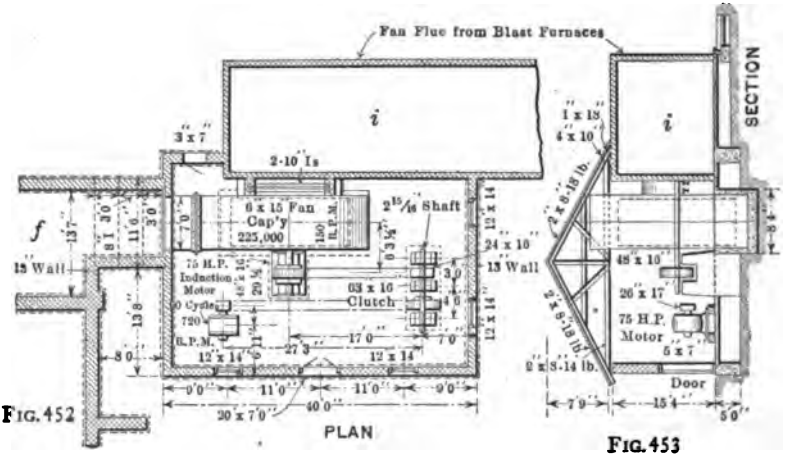

FIG. 452 FIG. 453

FIGS. 451 to 453.—Bag filtration, Murray, Utah.

The flue-dust is allowed to accumulate in a compartment to a depth of not more than 24 in., as with a greater depth there is danger of self-ignition. When it has attained the permitted thickness, it is ready to be ignited and heap-roasted. The compartment is cut off from the distributing flue and connected with the return-flue; the front doors are opened and glowing coal is spread over the surface to kindle the dust, which burns for 5 to 6 days. The heat generated by the oxidation of carbon and metallic sulphide is sufficient to sinter a large part of the dust sufficiently to permit its being handled by shoveling. It is shoveled into wheelbarrows and sampled at the same time by fractional selection. All laborers having to work with dust are required to take a shower

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bath after the work is over, to change their clothes, and are forbidden to eat their meals with dirty hands.

During 8 months of 1911 there were smelted 254,794 tons charge; there were recovered in the bag house 1124 tons fume, or 0.44114 per cent. of the weight of charge. Table 103 gives the distribution of metals in charge and fume, and Table 104 partial analyses of raw and sintered fume.

TABLE 103.—DISTRIBUTION OF METALS IN CHARGE AND FUME

	Ounces Au	Ounces Ag	Pounds Pb	Pounds Cu
Total metals in charge smelted...	17,319.854	3,472,681.90	50,079,080.00	1,834,355.00
Total metals in 1124 tons fume recovered.	16.754	2,948.35	638,861.00	1,295.00
Per cent. of metal in charge recovered in fume.	0.097	0.085	1.276	0.071
	Oz. per ton	Oz. per ton	Per cent. Pb	Per cent. Cu
Assay of the 254,794 tons of charge	0.068	13.629	9.827	0.36

TABLE 104.—PARTIAL ANALYSES OF RAW AND SINTERED FUME

	Ounces Au per ton	Ounces Ag per ton	Per cent. Pb	Per cent. Cu	Per cent. SiO_2	Per cent. Zn	Per cent. S	Per cent. As	Per cent. Fe
Assay and analysis of the 1124 tons raw fume recovered.	0.0149	2.62	28.42	0.058	2.9	3.2	5.2	36.6	5.1
Assay and analysis of sintered fume.	0.014	2.4	27.3	0.05	2.5	4.2	4.2	33.5	5.2

The cost of all operations of the bag house for 4 years and 8 months is given in Table 105.

TABLE 105.—RECOVERY OF FUME, ETC., AND COST OF, FOR ENTIRE OPERATION OF BAG HOUSE, JULY 7, 1907, TO FEB. 29, 1912

	Charge smelted	Fume recovered	Assay per ton fume recovered	Per cent. of charge-contents smelted	Value of metals recovered
Weight.....	1,666,857 tons	13,857,645 lb.	.....	0.415	.....
Gold.....	256,874.986 oz.	125.595 oz.	0.018 oz. per ton	0.049	$2,587.10
Silver.....	24,725,274.19 oz.	14,885.62 oz.	2.15 oz. per ton	0.063	8,028.25
Lead.....	354,693,161 lb.	4,492,288 lb.	32.4 per cent.	1.269	141,745.02
Copper.....	16,521,715 lb.	1,947 lb.	.....	0.0118	330.15
Total.....	.....	.....	.....	.....	$152,690.52

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Operating expense		Outcome	
Labor	$16,440.05	Treatment charge 6929 tons at $10.00	$69,290.00
Motive power	13,542.10	Operating cost	76,853.43
Supplies and repairs	46,871.28	Total cost	$146,143.43
Total	$76,853.43	Value of metals recovered	152,690.52
		Gain for 4 years and 8 months	$6,547.09

The statement in Table 105 shows an apparent gain from the installation of the bag house in 4 years and 8 months. But the figure does not include interest on original investment, nor a fair percentage for amortization. If these two things are included, as they should be, the apparent gain is changed into an actual loss, as shown in Table 106.

TABLE 106.—PROFIT AND LOSS OF OPERATION

Gain in 4 years and 8 months		$6.547.09
6 per cent. interest on cost of $127,194.89 for 4 years and 8 months	$35,614.57	
5 per cent. amortization for 4 years and 8 months	29,678.80	65,293.37
Net loss in 4 years and 8 months		$58,746.28

If the As_2O_3 in the fume were recovered, the loss of $58,746.28 would be reduced by $34,654.00, but there would still be a net loss of $24,101.28.

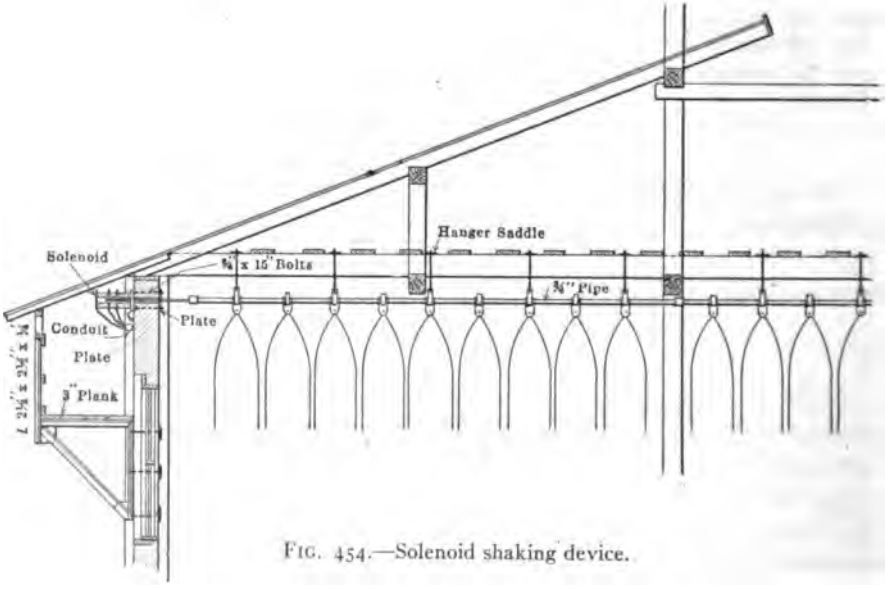

FIG. 454.—Solenoid shaking device.

210. Solenoid Bag-shaking Device.—The bags[1] in a silver-lead smeltery are usually shaken once or twice a day by means of a lever attached outside of

[1] Bennet, *Min. Eng. World*, 1908, XXIX, 633; *Eng. Min. J.*, 1908, LXXXVI, 451, 1009.

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the building to a pipe from which a row of bags is suspended. This has been discussed in §208 with the Murray bag house.

Shaking by hand has been replaced in one instance by the electric device shown in Fig. 454. In the bag house each row of bags is suspended from a ¾-in. gas pipe supported at 4-ft. distances by light rod-hangers which permit the pipe to have a pendulum motion in the direction of its axis. The pipe projects through the bag house wall at one side and ends in a soft-iron plunger which moves in a compound solenoid mounted on the wall. An intermittent current, transmitted alternately to each end of the solenoid, causes a reciprocating motion in the plunger and pipe which produces a wave-like motion in the bags and detaches the dust. The current is supplied through a single contactor in the fan house. The bag house has 84 rows of bags and therefore 84 solenoids;

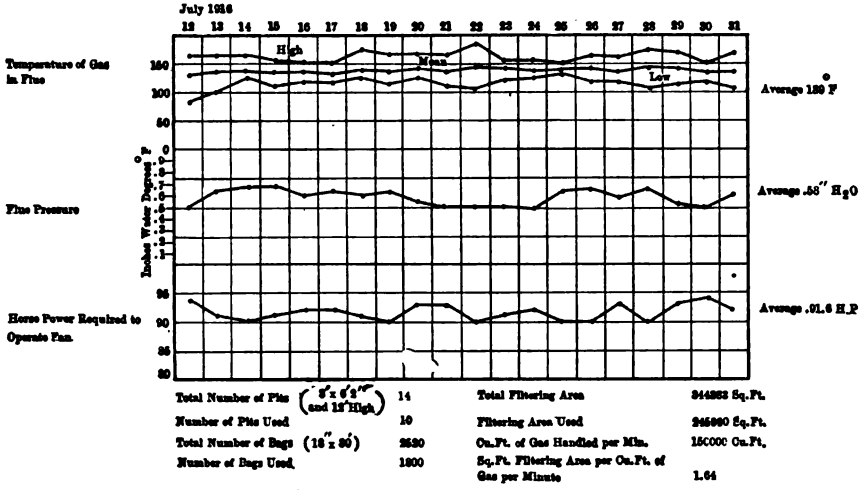

FIG. 455.—Bag house record.

each solenoid is placed in circuit by throwing in a knife-switch. The bag house record for July 1916 is given in Fig. 455. The bags are shaken whenever the gauges show a pressure above normal, usually about every 2 hr. A uniform flow of gas to each pit is obtained by keeping the difference in pressure between flue and pits as constant as possible.

211. Bag Filtration at Midvale, Utah.[1]—At this smeltery the smoke, both from the roasters and blast furnaces, is passed through the bag house before it is allowed to escape into the open. The gases from the roasting division, which

[1] Ingalls, *Eng. Min. J.*, 1907, LXXXIV, 527, 575.
Brinsmade, *Mines and Minerals*, 1907, XVIII, 216.
Benedict, *Min. World*, 1908, XXIX, 633.
Rice, *Mines and Methods*, 1909, I, 9.
Palmer, *Eng. Min. J.*, 1909, LXXXIX, 853.
Anderson, *Tr. A. I. M. E.*, 1914, XLIX, 570.
Austin, *Min. Sc. Press*, 1916, CXII, 746.
Private notes, 1917.

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contain more SO_3 than is permissible in bag filtration, are neutralized by treatment with ground lime and zinc oxide in the Sprague process; the gases from the blast furnaces are filtered direct. A sketch of the filtering plant is given in Fig. 456. The roasting division contains Wedge roasting kilns, *r*, pot roasters, *l*, and Dwight-Lloyd sintering machines, *g*. The gases from the Wedge kilns pass into flue *j*, which has at one end the lime-feed, *s* (see page 454); the pot roasters deliver their gases into flue *k*, which is provided with the lime-feed, *t*; both flues, *j* and *k*, terminate in the main flue, which also receives the gases from the Dwight-Lloyd sintering machines through

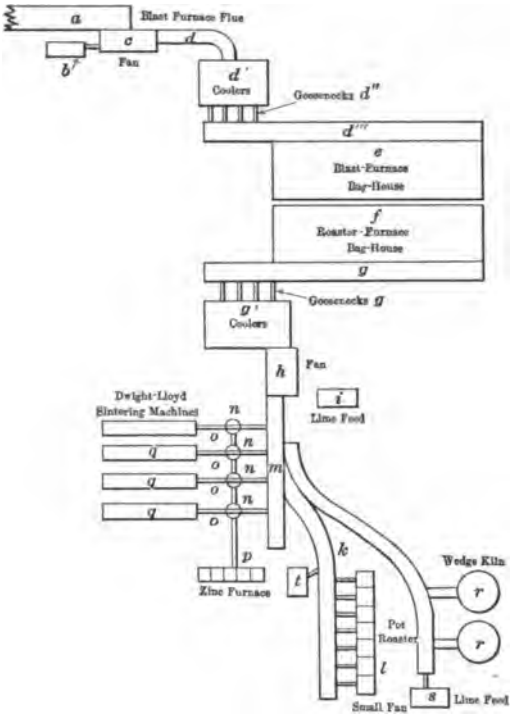

FIG. 456.—Filtering plant, Midvale, Utah.

branches, *o*, provided with plug-valves, *n*, which control the amounts of zinc oxide produced in the Wetherill zinc furnaces, *p*, from blende concentrates. The main flue enters the fan house, which has its lime-feed, *i*. The gases, neutralized as to SO_3, pass through coolers, *g'*, and goose-necks, *g''*, into the distributing flue, *g*, and thence into the roaster bag house, *f*.

The gases from the blast furnaces, 6000 cu. ft. per min., arrive in the blast furnace flue, enter the No. 14 Sirocco fan, *c*, which for safety is provided with the lime-feed, *b*, and then through flue, *d*, coolers, *d'*, goose-necks, *d''*, and distributing flue, *d'''*, into the blast furnace bag house, *e*.

The blast furnace bag house is shown in Figs. 457 and 458. It is 139 ft. 5 in. long, 23 ft. 2 in. wide, and 33 ft. 6 in. high. The lower part, dust chamber *B*,

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is enclosed by brick walls; the upper part, bag chamber *C*, is a wooden framework with studs, 2 by 8 in. at 35-in. centers, and 2 by 8-in. girts spaced for 8-ft. corrugated iron with 6-in. laps, which is nailed on the outer side. The inside is lined with 36-in. asbestos sheets, ⅛ in. thick, nailed with 1-in. laps. The inclined outlet flue, *D*, is of similar construction. Both studs and girts are 2 by 6 in. and spaced at 24-in. centers; three sides are lined with tongued and grooved boards, and the whole is encased with No. 22 copper-coated sheet-steel. It ends at the base of a self-sustained brick-lined steel stack, 16½ ft. inner diameter and 210 ft. high.

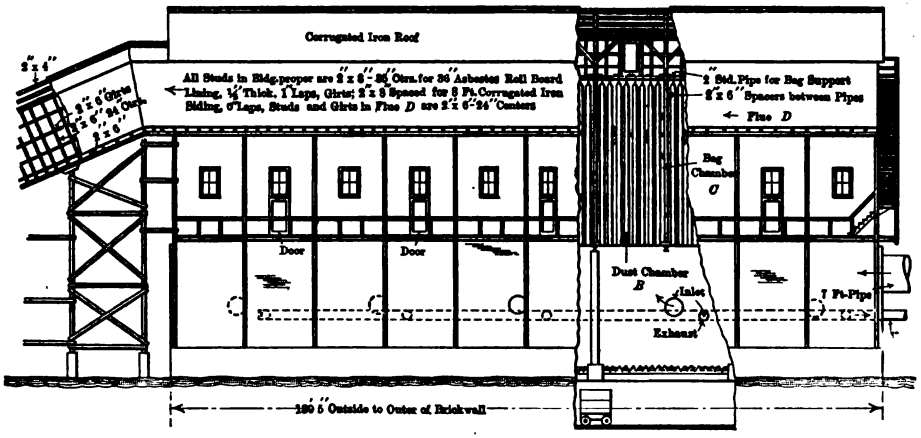

FIG. 457.—Blast furnace bag house, Midvale, Utah.

The thimble-floor, of 3/16-in. steel, carried by 12-in. I-beams, has 2400 thimbles, 11 in. in diameter and 10 in. high, which serve a corresponding number of bags, 12 in. in diameter and 34 ft. long, made of 42-in. canvas, double-sewed with 1-in. laps, wired at the bottoms and suspended at the tops from 2-in. pipes, which are kept in place with 2 by 6-in. spacers.

The bags have a filtering surface of 324,000 sq. ft., which, with 60,000 cu. ft. gas per min., corresponds to 5.4 sq. ft. area per cu. ft. gas per min.

The dust chamber is divided by four walls into five compartments or bays, each provided with inlet-pipe for admission of gas, and exhaust-pipe for creating a vacuum; both are controlled by disk-valves. The bottom of a dust chamber is V-shaped with iron sides placed at an angle of 50°. In the trough is a helical screw, which is connected by means of a clutch with a motor-driven counter-shaft, and delivers the dust to the discharge-opening of a bay, placed under the division wall, whence it drops into a car.

The blast furnace gases are collected in the main brick flue, from which a 12-ft. elbow leads them to a 12-ft. balloon flue, 127 ft. long (*a*, Fig. 456) ending in the fan house, *c*, with lime-feed, *b*. The gases enter the distributing flue through a 7-ft. pipe and from this enter the five compartments to ascend in the bags. The finely divided lime settles on the bags and forms a protective coating. Leaving the bags, the gases rise to the top of the chamber, are directed down-

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ward by deflectors into flue *D* (Fig. 458), through which they travel with a speed of 9 ft. per sec.

Once a shift the inlets into the dust chamber are closed and the exhausts opened which connect with the exhaust main, *x*, and through this with the balloon flue on the exhaust side of the fan. The pressure in the bags, usually 1 to 2 in. water, is replaced by a slight vacuum; the bags collapse and loosen the dust which has collected on them. About 5 minutes is allowed for the dust to drop into the dust chamber. The operation is repeated two or three times. This method of detaching the dust is not as efficient as shaking by hand or

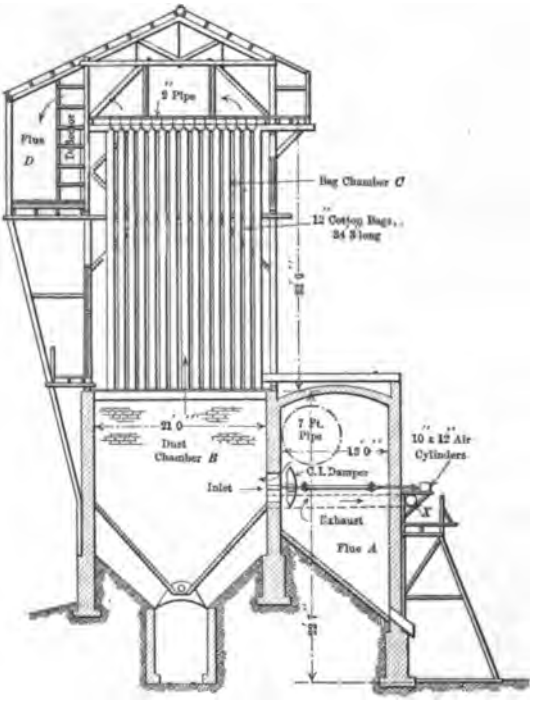

FIG. 458.—Blast furnace bag house, Midvale, Utah.

mechanical means, but it prolongs the life of a bag, as the alternate deflations and inflations are less severe on the filtering cloth. Once in several days the bags are shaken by hand in order to make up for the deficiency. Any dust accumulating on the V-shaped sides of the dust chamber is released by jarring with a hammer or a bar.

The bag house for the roaster gases has the same construction as that of the blast furnace division. The advantage of this construction, besides its low cost of $140,000, is that there is little condensation of moisture as compared with a brick building, and consequently less corrosion.

The flue-dust from the roaster gases carries 35 to 42 per cent. Pb, 9 to 16 per cent. As, and 10 oz. Ag per ton. It is briquetted, the lime furnishing the necessary bond, and goes to the blast furnace charge. The flue-dust from the blast furnace

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contains 18 to 22 per cent. Pb, 35 to 45 per cent. As_2O_3, and less than 1 oz. Ag per ton. It goes to the arsenic plant, and the residue from this back to the blast furnace. The As-content of the dust must be kept below 45 per cent., as otherwise the dust on the cars will tend to burn on its way to the arsenic plant. The dust in the hoppers of the chambers is kept cool by air circulation, but must be removed frequently, as otherwise it will ignite.

212. Sprague Process.[1]—This process aims to neutralize SO_3 in furnace gases by combining it with CaO and ZnO so that the gases can be filtered through bags without corroding the cotton or woolen cloth of which they are made. Lime is less effective than zinc oxide, but being the cheaper reagent is ordinarily used when the gases enter a flue; zinc oxide is introduced at a later stage to finish the work left uncompleted by the lime. Both $CaSO_4$ and $ZnSO_4$ do not attack cotton or wool. As the union of CaO and ZnO with SO_3 is satisfactory only at temperatures below 120° C., the gases have to be cooled before neutralization may be used.

Gases from lead blast furnaces do not contain a sufficient amount of SO_3 to affect filter cloths. Gases from lead-matte converters contain sufficient PbO during the slag-forming stage to neutralize any SO_3 which is formed; this is not the case during the blister-forming stage, hence they have to be diluted with false air before they can be filtered. Gases from roasting and blast roasting processes cannot be filtered without having been neutralized.

The places at which CaO and ZnO are ordinarily introduced at the tops of the flues carrying the gas-currents at Midvale, Utah, is shown in the plan-sketch, Fig. 456.

The lime-feeder of Midvale is shown in Figs. 459 and 460. The feed-hopper, 12½ by 9¼ in. and 3 ft. long, delivers burnt lime through a port (with opening regulated by a gate) on to an endless belt which discharges into a hopper attached to the top of the gas-flue. The lime is ground through an 80-mesh screen and slaked with sufficient water to make it fluffy, as in this form it combines readily with SO_3. As it has a tendency to pack, the feed-hopper receives per minute between six and seven vibratory shocks from an upright pendulum which is thrown first to one side and then to another by means of two pins on a triangular plate, pivoted at the center and rocked to and fro by a connecting rod. The rod is attached to a crank-pin placed eccentrically on a spur-gear, and this is driven by a pinion-wheel on one of the shafts of the endless belt. The apparatus feeds per hour 240 lb. lime to a roaster treating in 24 hr. 270 tons sulphide ore containing 35 per cent. S.

Zinc oxide is prepared by treating blende mixed with 30 per cent. fine coal on a Wetherill grate or similar grated furnace. The oxide-laden gases are drawn into the flue, where they become intimately mixed with the gases traveling

[1] Sprague, *Min. Sc.*, 1908, LVII, 53; *Min. World*, 1909, XXXI, 553.
Rice, *Mines and Methods*, 1909, I, 9.
Ebaugh, *J. Ind. Eng. Chem.*, 1909, I, 686; *Eng. Min. J.*, 1909, LXXXVIII, 1020.
Sprague, *op. cit.*, 1910, LXXXIX, 519.
Bocking, *op. cit.*, 1911, XCI, 895.

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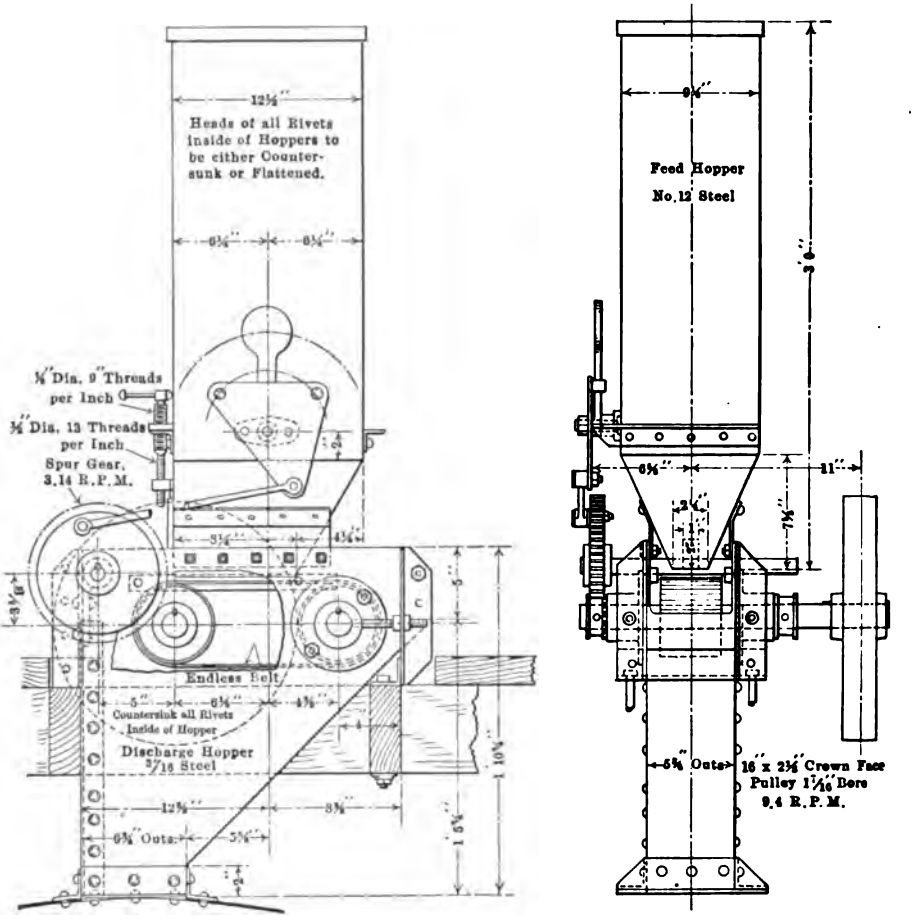
through it. The roasting charge is made up in such a manner that the roasted ore shall show an alkaline reaction of not less than 5 per cent.; this means that there must be present in the ore more than 5 per cent. available CaO, when it is analyzed for CaO, ZnO, and SO_3, and when the CaO and ZnO are figured as being present as $CaSO_4$ and $ZnSO_4$.



FIGS. 459 and 460.—Lime-feeder, Midvale, Utah.

213. Bag House, General.—Details of the bag houses at Murray and Midvale, Utah, have been given in §208 and 210. There are certain features which are found in every bag house, and others which are not contained in the preceding discussion; these are taken up here.

All bag houses have a horizontal partition, 12 to 16 ft. above the main floor, which divides the house into the upper filter chamber, 30 to 40 ft. high, and the lower dust chamber, 12 to 16 ft. high. The horizontal partition or thimble-floor, is usually made of sheet-steel, but sometimes of reinforced concrete as at Selby,

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Cal.[1] It carries from 3000 to 4000 thimbles[2] over which are slipped the bags to be tied with twine, wire, or strap-clamps. The dust chamber is divided into a number of compartments or cellars, each of which is connected through a port, provided with a closing valve, with the gas main. The partition of every third or fourth bay may reach to the roof, or only to the thimble-floor as do the others. In the former case, each filtering compartment has a separate sheet-iron stack to carry off the filtered gases; in the latter, the gases from the entire chamber are drawn off through a single flue, close to the roof, leading to the stack. The more quickly the gases are withdrawn, the better for the life of the bags.

A fan capable of moving from 150,000 to 250,000 cu. ft. gas per min. draws the gases from the furnaces through cooling and settling flues, 1000 to 1800 ft. in length, and delivers them at a temperature of about 70° C. to a distributing flue, from which ports extend into the cellars of the bag house. The pressure of the gas is from 1 to 2 in. water; higher pressures give greater filtering capacity, but shorten the life of a bag.[3]

The bags, 3000 to 4000 in number, are usually 18 in. in diameter and from 30 to 33 ft. in length. The filtration surface offered is about 3½ sq. ft. per cu. ft. gas per min.; it does not go below 2½ sq. ft.[4] and does not exceed 5 sq. ft. In regard to blast furnace tonnage, it is the usual custom to provide from 300 to 500 sq. ft. surface per ton charge smelted in 24 hr. At Midvale, Utah, there are provided 25,000 sq. ft. filtering surface per ton of fume collected per day.

The bags are made of cotton and of wool. Cotton cloth has from 30 to 40 strands per linear inch depending upon the character of the fume to be filtered. According to Iles,[5] cotton bags become slightly colored at 100° C., but are not weakened; slightly browned at 125° C. with strength appreciably altered; much browned at 150° C. and decidedly weakened so as to tear easily. If in contact with acid fume of the filtered gas at temperatures of 35 to 40° C., the cloth is rapidly attacked.

Woolen cloth, owing to the longer nap, is of a coarser weave than cotton, 20 strands to the linear inch; it retains its elasticity at temperatures as high as 120 and 130° C. and resists slightly acid fumes at 65 to 80° C. Gases with a temperature of 132° C. have been safely filtered. Wool is less attacked by acid than is cotton, and especially when the wool is allowed to retain its natural grease. When scoured wool is used in the manufacture of cloth, the grease is added later. Although a woolen bag at Murray (page 445) cost nearly 2.2 times as much as one of cotton, the longer life of the former more than makes up for the extra expense.

Alexander[6] made exhaustive tests on the critical temperatures of cotton and woolen bags. He found that the strength of cotton begins to decrease on ex-

[1] Bennet, *Eng. Min. J.*, 1908, LXXXVI, 451.
[2] Details: Alexander, *Tr. A. I. M. E.*, 1914, XLIX, 563.
[3] Anderson, *Tr. A. I. M. E.*, 1914, XLIX, 571.
[4] Anderson, *loc. cit.*, gives 1.6 sq. ft.
[5] "Lead Smelting," p. 210.
[6] *Tr. A. I. M. E.*, 1914, XLIX, 561.

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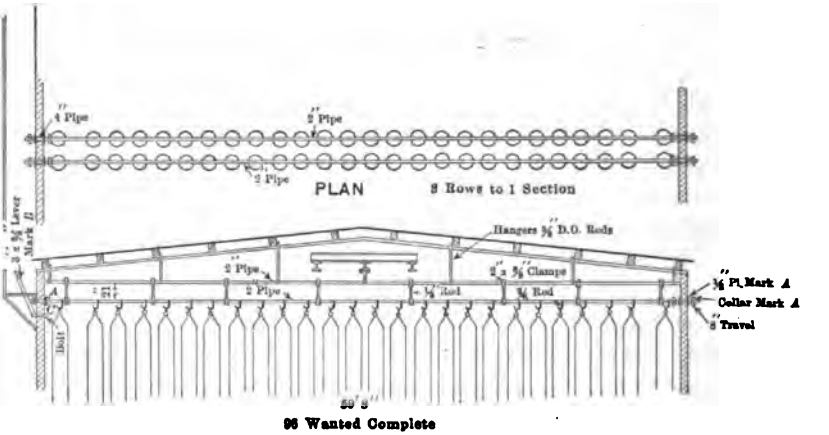
posure of 48 hr. to 100° C. (see Iles, above), whereas with a woolen bag the same takes place only after 96 hr. at 135° C. A 1-hr. treatment begins to show its effect on cotton at 126° C., and on wool at 188°. The following is his specification for woolen cloth: "The weight is to average 12 oz. per yard, and the tensile strength is to be not less than 21.5 lb. per linear inch. The test pieces are to be square, 3.5 in. on a side, and pull is to be against the warp. The fabric is to contain not less than 85 per cent. wool fiber, estimated by taking the difference between 100 per cent. and the sum of grease, dirt, moisture, burrs, and cotton fiber, and to be practically free from vegetable matter of all kinds. The weave is to be 22 ends by 20 picks per inch."

At Pertulosa, Italy,[1] 17,700 cu. ft. gas of 70 to 80° C. are filtered through 10 to 12 frames of 430 to 592 sq. ft. with vertical threads of asbestos. The collected fume is dislodged by releasing the tension of the threads and vibrating the frames. This filtering medium of L. B. Fichter appears to be better suited than wool for gases of higher temperatures (150 to 200° C.) and of greater acidity. However, asbestos is not as resistant to acid fumes as is generally believed.

Hanging and shaking of bags shows a variety of procedures. The methods used at Murray (page 445) and Tooele (page 422) for hanging represent two characteristic examples of modern practice.



FIGS. 461 and 462.—Bag-shaking device, Globe Works, Denver, Colo.

Shaking bags to dislodge adhering fume used to be done by hand. The men have to be protected against fumes and gas by rubber suits, helmets, and supplies of compressed air. Shaking by hand, while more effective than by any mechanical device, has been given up in most plants, as it is slow and a very disagreeable piece of work. The mechanical device in use at Murray has been given on page 445.

The shaking device of the Globe Works, Denver, Colo., is similar to that in use at Murray. In Figs. 461 and 462,[2] a row of bags is suspended by hooks

[1] *Min. J.*, 1907, LXXXI, 665.

[2] Alexander, *Tr. A. I. M. E.*, 1914, XLIX, 568.

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from a 2-in. pipe, which in turn is carried by hangers about 8 ft. apart and 21 in. long. The ends of a pipe pass through the walls in cast-iron spools with holes sufficiently large for the up-and-down movement caused by swinging on a 21-in. radius. The ends of a pipe carry collars on each side of the cast-iron spool set for an 8-in. stroke. One end of a pipe carries a lever which, when pulled quickly back and forth, gives the bags an up-and-down as well as a back-and-forth motion, and, on coming up sharply against the collars, produces a sudden jar which assists in dislodging the fume from the bags. After shaking, the pressure inside the bags is lowered by 0.15 in. water.

The method of Benedict[1] aims, in shaking, to give the bags a motion similar to the one at the Globe Works. It was in use at Midvale, Utah, and is found at present at Selby, Cal.,[2] the swing bars being carried instead of being suspended.

The reverse-current bag-shaking system at Tooele (page 422) and Midvale (page 452) has been discussed already.

214. Electric Precipitation.—*Cottrell Process.*—The precipitation of fume by means of electrostatic charges, though tried in a number of instances, became an industrial success only with the Cottrell process,[3] in which a high-potential unidirectional intermittent current is passed from discharge-electrodes (silent or corona or glow discharge) through an air-space to collecting-electrodes. The air becomes ionized, particles of the electricity of the sign of the discharge-electrode flow to the collecting-electrode. The suspended particles are bombarded by the ions which attach themselves to the particles and give up their charges to them. The particles gather sufficient electric charge to move under the force of the electric field, and travel toward the receiving-electrode and are deposited, while the gases pass off unaffected. The speed of travel is proportional to the potential between the two poles, and the conductivity of the air. The source of electricity is an alternating current, usually taken from a commercial line. It is transformed in a stationary transformer to the desired voltage, ranging from 15,000 to 50,000 volts, and then changed into an intermittent direct current by means of a special rotary contact-maker driven by a synchronous motor.

The air from furnaces usually contains sufficient SO_3 or H_2O to make it conductive. Should this not be the case, it may be necessary to inject a fine spray of water, as was the case with Howard,[4] during the slagging stage of matte-converting (§186).

In lead works the process has been applied at Trail, B. C., for blast roasting and blast furnace gases; at Garfield (§185–187) and Tooele, Utah (§188 and

[1] *Min. World*, 1908, XXIX, 633.

[2] Benedict, *Eng. Min. J.*, 1908, LXXXVI, 451.

[3] Cottrell, *J. Ind. Eng. Chem.*, 1911, III, 542; *Min. Sc. Press*, 1911, CIII, 255, 286; *Eng. Min. J.*, 1911, XCII, 763; *Tr. A. I. M. E.*, 1912, XLIII, 512, 755; *Smithsonian Report*, 1913, pp. 633–685 (Publication No. 2307); abstract, *Min. Ind.*, 1914, XXIII, 867–899; *Proc. Am. Inst. Elect. Eng.*, 1915, XXXIV, 625 (incl. bibliography).

Bradley, *loc. cit.*, p. 523.

Cottrell, *Eng. Min. J.*, 1916, CI, 385 (incl. bibliography).

[4] *Tr. A. I. M. E.*, 1914, XLIX, 545.

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189), for converter fumes; at Newark, N. J., for gases from silver furnaces; and a number of other places.

At Trail, B. C.,[1] the wire-discharge and pipe-collecting system (multiple treater), first used at Garfield, was installed in 1915 for the recovery of dust and fume from the blast roasting, smelting, and converting divisions. A side-view and cross-section of the multiple treater for blast furnace gases are shown in Figs. 463 and 464. The building is an elevated steel structure with central concrete flue and six treater-sections on a side (at present increased to nine). A treater-section has 32 grounded, cast-iron pipes, 5 in. apart, which are 13 in. outer and 12½ in. inner diameter (the largest practical size), and 18 ft. long. A pipe has a central No. 11 cow-chain (the No. 10 log-chain formerly used, stretched and filled with dust) which receives a current of 0.2 to 0.3 amp. at 550 volts. The plant treats per minute 160,000 cu. ft. gas, of a temperature of about 180° C., which travel with a velocity of from 5 to 7 ft. per sec. The clearance under these conditions is not satisfactory, as the temperature is too high, and the velocity too great. It is the intention to reduce both, the latter to 4 ft. per sec. The pipes are exposed to the air, which causes the outer sections of the pipe to be cooler than the inner. As this interferes with uniformity of work, the apparatus is to be inclosed.

The gases arrive from the blast furnaces in the central concrete flue. From this, ports lead into the hopper-shaped bottoms of the sections which act as receiving and distributing chambers. The dust adhering to the pipes is dislodged at stated intervals by tapping the pipes with a hammer; it collects in the steel hoppers, and is discharged frequently so as not to reduce the volume of the receiving chamber. The purified gases enter two balloon flues, which are combined into a single flue leading to the stack. It is the intention to connect each treater chamber separately with the main exit flue so as to permit close regulation of the speed of travel by means of dampers. The requirements for satisfactory work appear to be uniformity of volume, temperature, speed (4 ft. per sec. and less), and composition of gas, and of amount of suspended particles. Any change in these five conditions requires special adjustment. The fume collected in the treater is mainly lead sulphide with 60 to 70 per cent. Pb.

The work of Garfield has been noted on page 415.

At the Tooele plant, the multiple precipitator treats per minute 135,000 cu. ft. gases from the Dwight-Lloyd sintering machines in 880 pipes, 12 in. inner diameter and 15 ft. long; the speed of gas is 4.3 ft. per sec.; the temperature is governed by the amounts of arsenious oxide present.

The multiple treater of the Balbach Smelting and Refining Co., Newark, N. J., is small, as it is reserved for the gases from the silver furnaces (§288).

Attention may be called to the electric precipitation of fume at the works of the Raritan Copper Co., Perth Amboy, N. J. Here the anode mud from

[1] Guess, *Canad. Min. J.*, 1915, xxxvi, 37.
Ann. Report for 1905, *Minister of Mines*, Victoria, B. C., 1916, 5, p. 79.
Private notes, 1916.

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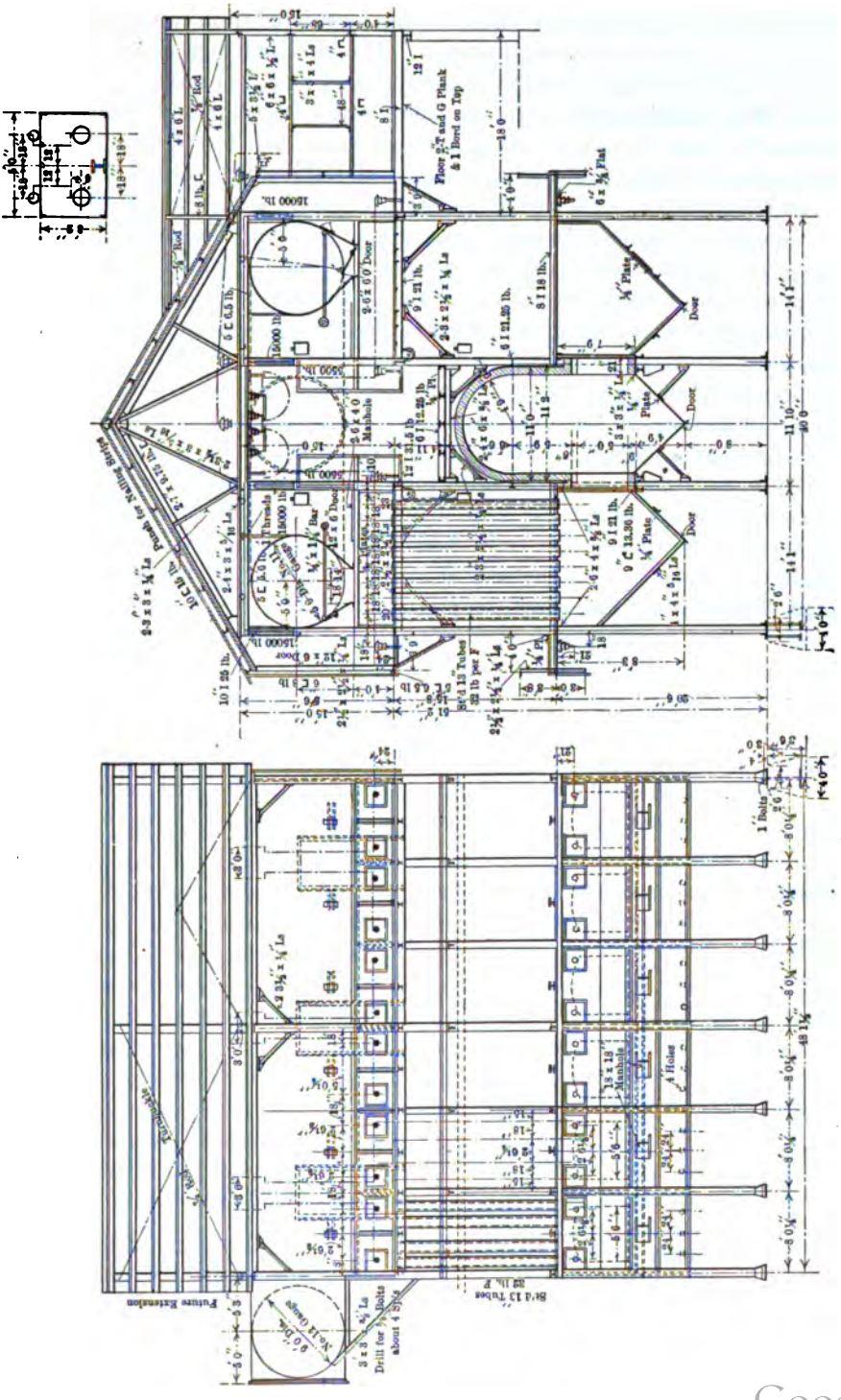

FIGS. 463 and 464.—Cottrell multiple treater, Trail, B. C.

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the electrolytic copper refinery, after having been freed from copper, is melted and fire-refined in reverberatory furnaces. The fume-laden gases are scrubbed and then drawn through two electric precipitators placed in series.[1]

215. Wet Condensation.[2]—In the collection of flue-dust and fume by wet methods, the gases have been drawn through water,[3] and forced through one or more horizontal filters[4] provided with sprays of water which carried down dust and condensed vapor. They have been drawn through wetted towers filled with coke, pebbles, brushwood, faggots, etc.,[5] or provided with horizontal staggered shelves,[6] or mounted with gable-shaped sheets of lead.[7] Sprays of water have been introduced into the current of hot gas in amounts sufficient to form mist or fog which would cool the gas and saturate floating particles so that they might settle when subjected to special cooling influences.[8] None of these methods have been sufficiently effective to warrant their use in lead plants in the United States. In Europe, on the other hand,[9] wet condensation is not uncommon.[10] The fact is that while wet methods can and do remove dust from gases almost completely, they are not efficient with fume. The Feld gas washer,[11] *e.g.*, has recovered in a lead plant 99 per cent. of the dust. They are common with iron blast furnaces for freeing gas from dust;[12] whereas dry filtration and electric precipitation are at present the only effective methods available for fume.

216. Treatment of Flue-dust.[13]—The question of the best manner of recovering values from flue-dust with little loss and cost is difficult to answer, as the treatment is governed by a variety of conditions.

The treatment of flue-dust from the ore-hearth has already been discussed (§70 and 72). There remains to be considered the dust from blast furnaces and from roasting, blast roasting, reverberatory, and converting furnaces. Blast furnace dust is likely to contain metallic sulphides and carbonaceous matter; dust from furnaces in which oxidizing operations are carried on is com-

[1] Hofman, "Metallurgy of Copper," 1918, p. 518.
[2] French, *Eng. Min. J.*, 1880, XXIX, 49, 84.
Iles, *School Min. Quart.*, 1895–96, XVII, 103.
[3] Eilers, *Tr. A. I. M. E.*, 1874–75, III, 310.
[4] Percy, "Lead," 1870, p. 442.
[5] Fallize, *Rev. Univ. Min.*, 1862, XI, 367.
[6] Iles, *loc. cit.*
[7] Richter, *Freiberg. Jahrb.*, 1889, p. 57.
Berg. Hüttenm. Z., 1890, XLIX, 129.
Eng. Min. J., 1890, XLIX, 196.
Doolittle-Jarvis, *Tr. A. I. M. E.*, 1910, XLI, 709.
[8] Ferraris, *Oest. Zt. Berg. Hüttenw.*, 1905, LIII, 455; *Eng. Min. J.*, 1905, LXXX, 783, at Monteponi, Sardinia.
Wynne, *op. cit.*, 1909, LXXXVIII, 602, at Neudorf, Harz Mountains.
[9] Heberlein, F., *Metall. u. Erz.*, 1913, X, 716.
[10] England: Roesing, *Zt. Berg. Hütten. Sal. W. i. Pr*, 1888, XXXVI, 103. Austrian Report of the "K. K. Arbeitsstatistisches Amt im Handesministerium" upon "Bleivergiftungen in Hüttenmännischen und Gewerblichen Betrieben, Hölder, Vienna, 1905–1913, pts. 1–8.
[11] *Met. Chem. Eng.*, 1912, X, 436, 1913, XI, 399.
[12] Hofman, "General Metallurgy," 1918, 864.
[13] Iles, *Eng. Min. J.*, 1886, XLI, 74, 93, 148.

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posed of oxides and is free from carbon. Many suggestions have been made and various methods tried for treating flue-dust.

1. *Wetting.*—To wet flue-dust and put it back into the furnace is of no use, unless it contains some anhydrous soluble sulphate, beause when dry it will simply be blown out again. The anhydrous salt takes up water, crystallizes, and binds together the particles of dust. The only sulphate of any importance in flue-dust is $ZnSO_4$, which at 720° C. is converted into $3ZnO.2SO_3$, and the latter at 767° into ZnO. Flue-dust containing from 1 to 2 per cent. Zn will harden easily when moistened. This is especially the case with the light-colored dust from roasting furnaces. The finely divided carbon in the blast furnace dust hinders this hardening to some extent. By mixing the two, satisfactory bricks can be obtained. An analysis of such a brick gave Livingstone: SiO_2 17.1, Fe 9.6, S 8.7, Zn 3.0, Pb 28.4, CaO 2.8 per cent. Another sample of bricked dust gave Dorr 3.04 per cent. ZnO. Treatment with water gave SO_3 0.35, Zn 0.04 per cent.; with ammonium sesqui-carbonate SO_3 1.70, ZnO 0.04 per cent.; with acetic acid SO_3 trace, ZnO 0.04 per cent.

The binding property of zinc sulphate was observed by the writer many years ago while roasting blende-bearing galena to extract by leaching the $ZnSO_4$ formed. After removing as much of the zinc as was possible, the residual ore, when dry, became so hard that it required strong blows with a sledge to break it. But the ore retained more than 1 or 2 per cent. Zn.

Hahn[1] found that flue-dust mixed with a solution of ferrous sulphate and formed into bricks (by hand), hardened sufficiently when sun-dried so that it could stand much handling. Perhaps an addition of slaked lime might prove useful, as it precipitates the ferrous oxide, which, on being exposed to the air, becomes peroxidized and forms hard lumps.

2. *Briquetting.*—A favorite method of compacting flue-dust is to mix it with lime and form it into bricks which when hardened are added to the furnace charge. Formerly from 8 to 10 per cent. of slaked lime was used and the bricks were made by hand. Cahen[2] gives an example of mixing flue-dust (with 44 per cent. Pb) and slaked lime. The bricks formed were smelted alone in the blast furnace with 74 per cent. of slag, and only 9.60 per cent. of the Pb charged was lost. More recent examples are those of Port Pirie, N. S. W.,[3] and Santa Fé, N. M.[4]

Considerable advances have been made in recent years in the bricking of flue-dust; they consist mainly in not bricking it alone, in the use of machinery, and in burning the bricks.

It has already been stated that blast furnace dust containing zinc sulphate will not harden as readily as that from roasting furnaces, on account of its carbonaceous matter, and for the same reason a binder, such as lime, added to it will

[1] *Min. Res. U. S.*, 1882, 344.

[2] "Métallurgie du plomb," Liège, 1863, p. 102.

[3] Howard, *Tr. Austral. Inst. Min. Eng.*, 1903, IX,[1] 106.
Delprat, *op. cit.*, 1907, XII, 15, *Eng. Min. J.*, 1907, LXXXVIII, 317, 517.

[4] Collins, *Tr. Inst. Min. Met.*, 1902–03, XII, 58.

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not act well. Hence more lime will be required for blast furnace dust than for roaster dust. Going a step further, the binding power of lime for roaster dust will be about the same as that for raw ore of the same degree of fineness and content of sulphur, as of course pyritic ores rich in sulphur, *e.g.*, concentrates, cannot be bricked, because upon heating they will fly to pieces. If, then, ore crushed to pass a limiting sieve of say 2 meshes to the linear inch be mixed in with fine ore and flue-dust, the additional advantage is gained of the angular shape of the crushed ore, which will require less lime for binding and will dry more satisfactorily on account of the mass of capillary spaces between the crushed dry ore, the moist flue-dust, and the cream-like binder. When the brick is being dried, either in the open air under sheds as at El Paso, Texas,[1] or additionally in a heated chamber, it will give up its water readily and on account of its porosity shrink without cracking. When charged into the blast furnace, part of it will again fall to pieces, but to a much smaller extent than if the flue-dust had been bricked alone with a high percentage of lime. If the dried brick, before charging into the blast furnace, is burned in a kiln, it will, when the temperature has been raised to the point where carbon and sulphides begin to roast, burn readily, and the heat generated will cause the lead or matte to cement together the single particles of a brick to a solid mass. This gives the explanation for the present practice of bricking fine silicious ores with from 2 to 5 per cent., or on the average 3 per cent., of slaked lime and 20 per cent. of more of flue-dust, to frit together the single particles when exposed to heat. With ores having a clayey gangue, or with roasted ore or roasted matte, no addition of flue-dust is necessary, if the bricks be kiln-burned; they will frit together, forming solid bricks and often as much as 50 per cent. of the S of the raw material will have been eliminated. With silicious ore free from sulphur, from 2 to 3 per cent. fine coke is added to the mixture to furnish the fuel necessary for burning in the kiln.

The bricks consisting of silicious ore and flue-dust weigh about 8 lb. apiece, those from roasted sulphide ores or matte about 12 lb.

In order to brick ores satisfactorily and cheaply with a small percentage of lime, it is necessary to subject them to pressure, which requires machinery. Of the different forms of brick presses which have been used, the Chisholm, Boyd, and White Mineral Press[2] is the machine that has outlived the others in lead plants. The main parts of this machine, shown in Fig. 465, are a roller mill and a molding disk. The pan of the mill is 7 ft. in diameter and is lined in the path of the rollers with chilled iron plates; the rollers are 48 in. in diameter, have a 10-in. face, are provided with removable chilled iron rings, and weigh 5000 lb.; a cross-beam keyed to the driving spindle carries a plow at either end which turns over the bricking mixture from both sides of the rollers into their paths. The mold-disk is a cast-iron plate, 5 ft. in diameter and 2½ in. thick, having two rows of circular holes or molds (4 in. in diameter at the top and

[1] Vail, *Eng. Min. J.*, 1914, XCVIII, 465.
Easter, *Tr. A. I. M. E.*, 1915, LII, 721.

[2] Chisholm, Boyd, and White Co., Chicago, Ill.

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slightly larger at the bottom), pivoted at the center. Its movement is effected by a radial arm journaled at the center and attached at the outer end to a pitman connected with a crank-gear. The arm has a round steel pawl, which drops into one of the holes, and two plungers. As it is pulled forward, the pawl draws the disk around 15° and the pitman forces the two plungers into the molds, pushing out a pair of briquettes on to a slow-moving endless belt. When the crank-gear again reaches its first position, the pawl drops into the next hole, and the plungers force out a second pair of briquettes and so on. The movement of rollers and disk is so adjusted that the rollers will travel 20 times over a pair of molds during their passage through the pan, thus firmly compressing the flue-dust mixture.

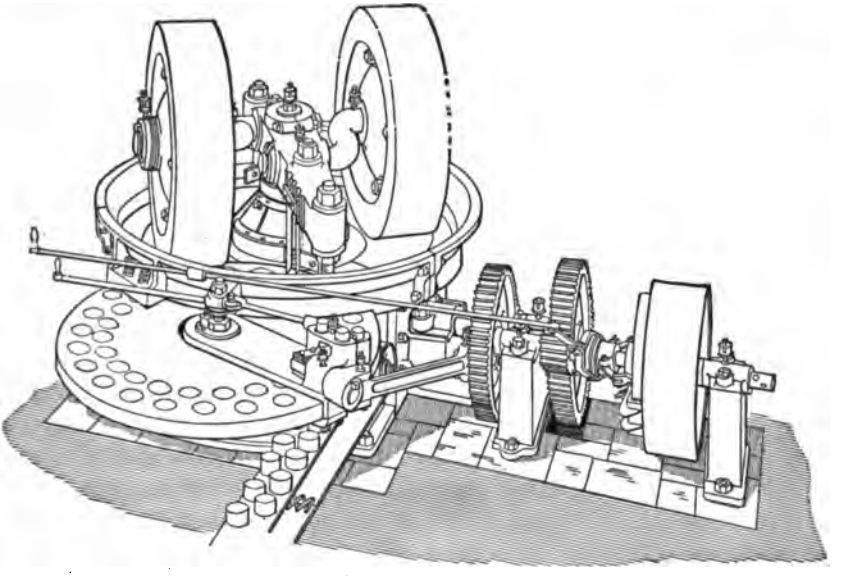

FIG. 465.—Chisholm, Boyd and White press.

The machine occupies a floor-space 9 ft. 6 in. by 11 ft. 6 in., and is 6 ft. 2 in. high; it weighs 27,000 lb., requires a 28-h.p. engine, and has a capacity of 4200 bricks (4 by 2½ in.) per hr. It requires two men at the pan, from two to four carriers to remove the bricks, and one foreman.

The green bricks received from a machine are loaded on brick trucks, and then piled up, rarely in racks, commonly on a covered floor, to a height of about 10 ft. and are air-dried for from 2 to 3 weeks. Drying chambers are of rare occurrence with Western smelting works, although the brick from them are much harder than if simply air-dried; with smelting works located in a less dry atmosphere the drying chambers may be necessary, especially during the cold season.

The burning of the brick is carried on in the ordinary up-draft kilns common for burning red-brick. The brick are set up in arches about 30 courses high, the arches being 20 ft. long and the open portion about 9 courses high. Six or more arches are built side by side. A kiln 80 ft. long, 10 ft. wide, and 20 ft. high will

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hold about 100,000 bricks. Fires are built in the arches, cordwood being commonly used. It takes from 3 to 4 cords to fire a kiln. The kiln begins to burn after a few hours of firing, which must then cease. It burns for 2 or 3 days and is pulled down after it has been given some time to cool. Burning briquettes in kilns was in operation in 1905 at Broken Hills, N. S. W.[1]

The bricking of flue-dust, which with hand-labor and a large percentage of lime cost formerly about $2 a ton, has been reduced by the use of machinery to from $0.20 to $0.25 a ton, excluding the cost of the lime.

Lime has probably to-day replaced all other binders. Harbordt[2] substituted clay, Church[3] used pan slimes, others have been successful with cement. Very satisfactory brick have been made by Carpenter[4] with the waste molasses of sugar plantations. Its composition is shown by the following table[5] in percentages:

Name of plantation	Moisture	Ash	Sucrose $C_{12}H_{22}O_{11}$	Albuminoids	Glucose $C_6H_{12}O_6$
Calumet	25.09	7.55	25.34	1.97	29.20
Calumet	26.15	9.35	26.02	2.40	28.93
Calumet	25.30	7.84	25.92	2.49	30.07
Calumet	26.09	7.01	25.46	2.30	31.31
Magnolia	30.37	9.34	27.65	1.92	21.12

The ash is principally lime and contains some potash. According to Stubbs,[6] the fuel value is 37 per cent. of that of average Pennsylvania bituminous coal (H_2O 1.68, V. H-C. 28.43, F. C. 64.05, ash 5.83; calorific power 3577 Cal. per lb. or 14,200 B.t.u.), 1 lb. molasses equals about 0.37 lb. coal.

3. *Fusion.*—Previous to the improvements made in bricking, the common method of treatment was to melt the flue-dust in a special reverberatory furnace or in the fuse-box of a hand roasting furnace, where it was added in quantities of, say, 100 lb. to the roasted ore. It was charged before the ore was drawn from the roasting-hearth, and was thus covered by the roasted ore, so that little metal was carried off mechanically. It helped to sinter the charge, but there was considerable loss in lead, although little in silver. Murray found that removing the finely divided carbon by washing assisted the fusion.

A charge of 150 tons of blast furnace flue-dust taken from the first 400 ft. of the dust-chamber of the Globe Smelting and Refining Co. gave, according to Iles:[7] SiO_2 16.6, Fe 8.6, CaO 5.2, Pb 27.6, Zn 3.0, S 6.3 per cent., Ag 20.0, and Au 0.16 oz per ton. After fusing, it had changed to SiO_2 21.0, Fe 12.4, CaO 5.2, Pb 37.8, Zn 3.2, S 6.4 per cent., Ag 34.0, and Au 0.26 oz. per ton.

[1] Greenway, *Eng. Min. J.*, 1905, LXXIX, 73.
[2] Private communication, 1891.
[3] *Eng. and Min. J.*, 1885, XL, 124; *Tr. A. I. M. E.*, 1886–87, XV, 611.
[4] *Op. cit.*, 1900, XXX, 768.
[5] United States Department of Agriculture, Bulletin No. 18, tables 53 and 53 bis, 25.
[6] *The Louisiana Planter and Sugar Manufacturer*, July 13, 1805, 25.
[7] Private communication, 1897.

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In melting flue-dust in the reverberatory furnace, some lead bullion and matte are often formed beside the slag. The three products of a charge gave Iles the following values:

Product	Ag, oz.	Au, oz.	Pb	SiO_2	Fe	CaO	S	Cu	As
Slag	21.0	0.08	18.4	22.4	12.4	2.0	1.6	.....	.....
Matte	160.7	1.54	46.8	.....	.....	.....	.....	15.4	1.4
Lead bullion	450.9	6.12	.....	.....	.....	.....	.....	.....	.....

4. *Blast Roasting.*—At Trail, B. C., the flue-dust is removed mechanically from hollow flues and transferred to iron-plate cylinders, 7 ft. in diameter and 10 ft. high, lined with 4 in. of brick and covered with removable conical hoods provided with doors and gas-escapes. A cylinder is closed at the bottom with a grate, and provided with under-grate blast. It thus resembles a Huntington-Heberlein pot adapted for blast roasting flue-dust. The fire is made on the grate, the blast is started, and the flue-dust charged gradually. The cylinder is then filled in about 3 days, in which time it furnishes from 17 to 20 tons blast roasted dust. This is hard on the outside, and resembles on the inside the burnt dust of the cellar from a bag house. When the charge has been roasted, the hood and cylinder are removed by means of a crane, and the dust is transferred to bins from which the blast furnace charges are made up.

5. *Addition to Blast roasting Charge.*—Since blast roasting has become such an important auxiliary process to blast furnace smelting for the preparatory treatment of sulphide concentrates, the practice of adding flue-dust to the blast-roasting charges has become very common. Though it is true[1] that flue-dust is not desired in this connection, less so in a Huntington-Heberlein pot than in a Dwight-Lloyd machine,[2] it furnishes a quick and cheap method of handling this material. It cannot be more unwelcome than flotation concentrates, which are being recovered by smelteries in increasing amounts, and have to be blast roasted. The fact is that blast roasting practice will have to be adapted to meet these new demands.

217. Losses and Cost.—The losses in smelting are due to slagging or to particles of metal being carried off in by-products and not recovered. As has already been stated (§131), ore-slags should not contain over 0.75 per cent. Pb and ½ oz. Ag to the ton with 300-oz. bullion; but they often contain over 1 per cent. Pb and about 1 oz. Ag, and are considered satisfactory. Special causes, such as the presence of foreign substances having a deleterious effect, may make slags run still higher. One of the main sources of loss is that matte remains dissolved in the slag; often, also, pellets of matte do not settle out completely. This, according to Hering,[3] is sometimes due to the oxidizing action

[1] Palmer, *Tr. A. I. M. E.*, 1914, XLIX, 511.

[2] Norton, *op. cit.*, 1914, XLIX, 488.

[3] *Oest. Zt. Berg. Hüttenw.*, 1893, XLI, 238.

Schertel, L, "Studien über einige Verlustquellen des Blei- und Kupfer-Hochofen Processes," Freiberg, 1911.

Wanjukow, *Metallurgie*, 1912, IX, 1.

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that certain slags have, especially if they contain Fe_3O_4. These acting upon the sulphides set free SO_2, which adheres to pellets of matte and hinders them from settling.

The quantity of by-products (speiss, matte, flue-dust) formed has an important influence on the output of Pb and Ag, as they have to be retreated, which causes an unavoidable loss in metal. It is difficult, therefore, to give an average figure of the Pb and Ag recovered in smelting in the blast furnace. It may be said that a total loss of from 6 to 7 per cent. Pb represents very good work with clean ore, the lead assays being made in the dry way. As long as the loss does not exceed 10 per cent., the work may be considered satisfactory. The report to the Treasury Department of the United States by Hofman[1] dealing with the loss in Pb prevailing at smelteries treating foreign ores, has shown that the loss in Pb varied from 5 or 6 to as much as 20 per cent. A decision of the Treasury Department[2] allows the works of the American Smelting and Refining Co., at Maurer, N. J., a loss of 20 per cent., and those at El Paso, Texas, 19.9 per cent. These large losses are caused by the treatment of lead ores rich in As and Zn.

With silver[3] the yield is generally from 98 to 99 per cent., the silver-assay being made in the dry way and not corrected. With losses of Pb reaching 20 per cent., the yield in Ag is lower than 98 per cent.; the decreased yield of Ag, however, does not correspond with the increased loss of Pb, but is a great deal less.

The yield in Au is 100 per cent., the Au being paid to 0.05 oz. per ton; often there is a *plus* gold.

The cost of smelting[4] in the Rocky Mountain district, where blast roasted ore is largely used in preparing the charges, ranges from $2.25 to $2.50 per ton charge, provided the plant is large and has all of its furnaces in operation. If converting blast furnace matte be included, the cost may rise to $4.00 per ton charge. With a small plant the cost of treatment is approximately double that of a large plant. Of the total cost,[5] labor accounts for about 23 per cent.; coke, 40; coal, 5; limestone for coke-ash, 5; maintenance and repair, 5; delays caused by accidents, strikes, etc., 5; flue-dust recovery, 2; administration, 7 per cent.

[1] *Eng. Min. J.*, 1911, XCI, 1193.
[2] *Op. cit.*, 1913, XCV, 532.
[3] Iles, *Eng. Min. J.*, 1899, LXVIII, 307, 340, 367.
Godshall, *op. cit.*, 1900, LXIX, 43.
[4] Finlay, *Min. Sc. Press*, 1908, XCVI, 22; *Eng. Min. J.*, 1908, LXXXV, 165.
Adkinson, *op. cit.*, 1908, LXXXV, 992.
Brownlee, *loc. cit.*, 1110.
Grabill, *op. cit.*, 1908, LXXXVI, 73.
Ingalls, *loc. cit.*, 315; *Min. Ind.*, 1908, XVII, 608.
Guiterman, *Eng. Min. J.*, 1908, LXXXVI, 960.
[5] Austin, *Min. Sc. Press*, 1914, CIX, 170.

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CHAPTER X

DESILVERIZATION OF LEAD BULLION

218. Introduction.—The final separation of silver and lead is usually accomplished by the process of cupellation (§273). Up to the introduction of the Pattinson process (§219) all argentiferous lead was cupelled; but this has many disadvantages, prominent among which are the cost and the loss in metals, the limit being very soon reached where the separation of lead and silver ceases to be economical. This is the case with lead bullion assaying about 30 oz. Ag per ton. Below this amount, the silver recovered will hardly pay for the labor, fuel, and material required, the loss in metal, and the inferior grade of lead obtained from the reduction of litharge. It becomes necessary to concentrate the silver into a smaller amount of lead before cupelling. The process of Pattinson, with its modifications, and of Parkes do this successfully.

The progress made in desilverizing in the dry way during the last 80 years is well illustrated in Table 107. The figures published by Hermann[1] are derived from the actual working results obtained at the Ems Smelting and Refining Works, Prussia, where the three processes were used one after another. To these have been added the amount of lead to be cupelled and the traces of gold.

TABLE 107.—COMPARISON OF CUPELLATION, PATTINSON, AND PARKES PROCESSES

	Cupellation process	Pattinson process	Parkes process
Cost	3.0	1½	1.0
Amount of lead to be cupelled	100.0	13.0	5.0
Loss in lead and silver	6.0	2.0	1.0
Impurities remaining in the lead, per cent.	0.2	0.05	0.015
Silver remaining in the lead, ounces per ton	0.73	0.58	0.17
Traces of gold	Lost	Lost	Recovered

At Tarnowitz, Silesia, Roesing[2] experimented about 1890 with the oxidation of lead in a basic-lined converter. He worked with charges of lead weighing 13,200 lb. and assaying from 12.4 to 196.3 oz. Ag per ton; the fumes which he collected assayed 75 per cent. Pb and 2.5 oz. Ag per ton. He also refined, by a few minutes blowing, desilverized zinc-bearing lead.

[1] *Berg. Hüttenm. Z.*, 1883, XLII, 382.

[2] *Rev. Univ. Min.*, 1892, XVII, 110; *Berg. Hüttenm. Z.*, 1892, LI, 102; *Eng. Min. J.*, 1892, LIII, 431; *Stahl u. Eisen*, 1892, XII, 370.

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In 1878 Keith[1] started experiments with the electrolysis of lead bullion with lead acetate as electrolyte, and later built a plant which, however, was not successful.

In 1906 Tammasi issued a pamphlet[2] in which he discusses his apparatus for the electrolytic refining of lead, which consists of a revolving metal disk half-immersed in an electrolyte made up of solutions of acetates of lead and potassium. The spongy cathode lead is to be scraped off mechanically, washed, compressed, and melted, and the anode mud worked by some process. Nothing more has been heard about this process.

Later, about 1902, Betts electrolyzed lead bullion with lead fluosilicate, acidified with fluosilicic acid, as electrolyte. This process is considered in §290.

The task of modern desilverizing (or refining) works, is not only to separate effectively and cheaply the precious metals from lead, but also to make out of a lead bullion containing from 95 to 98 per cent. Pb a refined lead of not less than 99.9 per cent. Pb and salable products of the impurities contained in it, such as Cu, Sn, Bi, As, and Sb.

In the following pages the four standard processes for desilverizing lead bullion will be discussed in the order: Pattinson process, Parkes process, Cupellation process, Betts process. Cupellation follows the processes of Pattinson and Parkes, as it has ceased to exist as an independent process, having become an auxiliary to the other two.

PATTINSON PROCESS

219. Pattinson Process, General.[3]—This process is based upon the fact, discovered by Hugh Lee Pattinson in 1833, that if low-grade lead bullion is melted and then cooled to its freezing-point, crystals of lead will separate which are much poorer in Ag than the original lead. If they are removed and the process is repeated, always adding fresh lead of the same tenor in Ag, a large quantity of market lead low in Ag will result, and a small amount of enriched lead ready to be cupelled. By the repeated meltings and crystallizations necessary, many of the impurities will be collected in drosses, and the market lead become correspondingly purified.

The explanation of this phenomenon, which formerly was not understood, became clear with the application of the laws of solution. The freezing-point curve of the alloy series Pb–Ag has been given in Fig. 11. With Pb solidifying at 327° C., the eutectic of Pb–Ag with 2.5 per cent. Ag is seen to freeze at 300° C. If a low-grade fused alloy of Pb and Ag is considered as a dilute solution of Ag

[1] *Eng. Min. J.*, 1878, XXVI, 26, 59; 1882, XXXIII, 292; 1883, XXXVI, 372. *Tr. A. I. M. E.*, 1882, X, 312; 1884–85, XIII, 310.

Hampe's criticism, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1882, XXX, 81; *Eng. Min. J.*, 1882, XXXIII, 144.

[2] "Electrolytic Process for the Extraction, Separation and Refining of Metals," Paris, 1906; *Min. Ind.*, 1906, XV, 543.

[3] Percy, "Metallurgy of Lead," London, 1870, p. 137.

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in Pb, the pure solvent, Pb, when the alloy cools, will separate and, being heavier than the remaining mother-metal, will sink to the bottom. It will be noted that the range of temperature between the freezing-point of Pb (327° C.) and of eutectic (300° C.), that is, the range in which crystallizations must be carried on, is only 27° C. The result is that the separating crystals are mechanically contaminated with mother-metal, and therefore silver-bearing. The higher the Ag-content of the original lead, the less perfect will be the separation of liquid mother-metal and crystallized lead. There will be reached a point beyond which the enrichment of the liquid lead cannot be carried. Table 108 by Reich[1] shows that process has to stop when the lead bullion

TABLE 108.—LIMIT OF ENRICHMENT OF AG IN PATTINSON PROCESS

Ounces Ag per Ton		
In the molten lead before crystallization	In the crystals	In the liquid lead
205.33	113.74–135.91	298.95
213.49	92.75–109.08	313.83
281.34	119.58–198.33	422.91
288.16	113.74–181.99	446.24
420.57	198.91	560.57
609.57	586.53	659.15
615.15	503.99–646.31	655.65
643.40	645.15	660.32

assays from 600 to 650 Ag per ton. Practical experience has proved that the liquid lead cannot be enriched to more than 450 or 500 oz. Ag per ton. Further, the nearer the Ag-content approaches 650 oz. per ton, the smaller become the crystals, and the more difficult is it to drain off the liquid lead, especially as this tends to solidify at the same time as do the crystals.

In working the process, it has been found that Au followed the Ag. This might be expected from the freezing-point curve of the Pb–Au alloy series given in Fig. 12, and from the fact that Au and Ag form solid solutions. At Eureka, Nev.,[2] in 6 months of 1881, there were treated 15,993 tons lead bullion containing 541,299.41 oz. Ag and 16,794.50 oz. Au with a loss of 0.3 per cent. Ag and 1.9 per cent. Au; in 1882 the losses were 0.7 per cent. Ag and 1.0 per cent. Au.

The metal Bi shows a similar tendency, but it does not follow Ag as closely as does Au.

Freezing at 269° C., it forms[3] with Ag, freezing at 961.5° C., an eutectic with 97.5 per cent. Bi solidifying at 262°, and at the eutectic line a solid solution of Bi in Ag to the extent of about 5 per cent. From the Pb–Bi curve, Fig. 13, it is seen that Pb and Bi are mutually soluble at eutectic line to an extent of

[1] *Berg. Hüttenm. Z.*, 1862, XXI, 251.

[2] Letter of R. K. Morrison, Supt., March 7, 1901.

[3] Petrenko, *Zt. anorg. Chem.*, 1906, L, 138.

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about 10 per cent., but there is no indication of this in the Pb–Ag curve, Fig. 11, and the Pb–Au curve, Fig. 12.

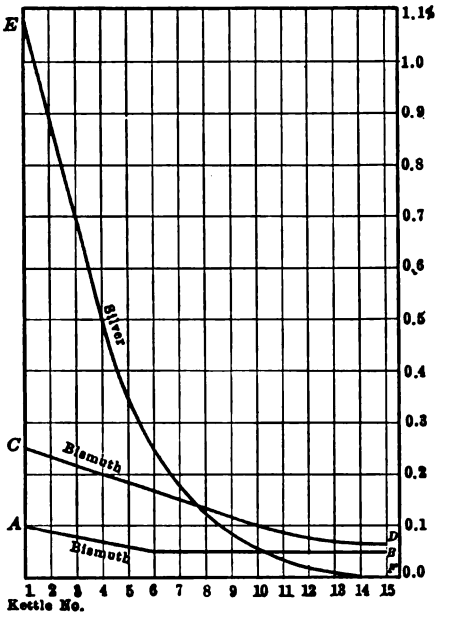

FIG. 466.—Decrease of Ag and Bi in Pattinsonizing.

Junge[1] found at Freiberg, Saxony, that the richer the Pb is in Ag, the more easily will the Bi be concentrated in it. Thus, lead bullion with 145.83 oz. Ag per ton and 0.06 per cent. Bi gave rich lead with 583.32 oz. Ag per ton and 0.18 per cent. Bi, and market lead with 0.44 oz. Ag per ton and 0.02 per cent. Bi. When the lead bullion was richer in Bi, the Ag remaining the same, the rich lead retained more Bi (0.29 per cent.), and this was also the case with the market lead (0.054 per cent.).

How slowly the Bi-content of the crystallized lead diminishes in Pattinsonizing, if compared with the Ag, is shown in Fig. 466. The curve *A B* refers to a lead bullion containing 0.1 per cent. Bi, the curve *C D* to one with 0.25 per cent. The decrease of Bi in a series of crystallizations carried out with the method by thirds is shown in Table 109; the figures indicate the percentages of Bi.

TABLE 109.—BEHAVIOR OF BI IN PATTINSON PROCESS

	Per Cent. Bi													
No. of kettle	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Lead before crystallizing.	0.242	0.205	0.200	0.203	0.183	0.173	0.162	0.137	0.132	0.110	0.105	0.088	0.064	0.063
Resulting crystals.	0.235	0.207	0.204	0.208	0.176	0.158	0.157	0.128	0.115	0.112	0.085	0.074	0.070	(a)0.054
Resulting liquid lead.	0.290	0.247	0.243	0.242	0.181	0.217	0.212	0.183	0.174	0.140	0.130	0.101	0.092	0.065

(a) Market lead.

The behavior of Bi in Ag-free Pb is taken up with the Tredinnick process in §224. Of the metals commonly found in lead bullion, Sb, Bi, and Ni are concentrated in the liquid lead; As in the crystals; Cu which has not been removed with the dross, remains equally distributed in the two products.

The process of concentrating the silver in a small amount of lead may be conducted according to two systems, called the METHOD BY THIRDS and the METHOD BY EIGHTHS. In the first of these systems two-thirds of the

[1] *Freiberg. Jahrb.*, 1895, 3; *Min. Ind.*, 1896, v, 397.

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lead contained in the kettle is withdrawn in the form of crystals, while one-third remains behind as liquid lead. The crystals will then be about half as rich in Ag, and the liquid lead about twice as rich, as the original bullion. In the second system the bullion in the kettle is divided into seven-eighths crystals and one-eighth liquid lead, and the Ag content of the crystals is approximately one-third as much, and of the liquid lead approximately three times as much, as that of the original bullion. The latter method is, therefore, to be applied to very low-grade bullion. Stetefeldt[1] tried to find a general mathematical formula which would show the proportions in which leads of different contents in Ag should be divided to attain, with as few crystallizations as possible, a market lead of a certain tenor in Ag and an enriched lead. In practice the two methods by thirds and by eighths have become standards, especially the former. A variation of the method by thirds, the one with intermediary crystals,[2] aims to reduce the number of crystallizations. The contents of the kettle are divided —two-thirds crystals and one-third liquid lead—and the crystals taken out. The liquid lead, however, undergoes directly a second crystallization, so that intermediary crystals and final liquid lead will result. Thus the original lead is divided into $\frac{2}{3} = \frac{6}{9}$ normal crystals, assaying one-half as much as the original lead; $\frac{2}{9}$ intermediary crystals, assaying the same as the original lead; and $\frac{1}{9}$ liquid lead, four times as rich as the original lead. The method has, however, been abandoned where it was tried, as it complicated the process, and as considerable amounts of slightly enriched leads had to be kept on hand.

To carry out Pattinson's process successfully the lead bullion must not be very impure or run too high in Ag, and a sufficient amount must be used to permit the careful regulation of temperature necessary. All the foreign metals contained in the lead interfere with the crystallization and the effectual separation of the liquid lead from the crystals. Ordinary lead can be sufficiently purified by poling (§116) and removing the dross that collects on the surface; if Sn, As, and Sb are present to any extent, the lead has to be softened (§241) at a bright red heat before the crystallization can proceed.

220. Original Pattinson Process.—The plant consists of a set of from 8 to 15 spherical kettles (with the method by thirds), built closely together in a row. The kettles hold from 6 to 15 tons of lead; the smallest permissible quantity is $2\frac{1}{2}$ tons. Each kettle has a separate fireplace, so constructed that the flame shall pass beneath and behind the kettle, thence into a flue encircling it, and finally into the chimney, which has a damper to regulate the draft. The details of the construction are the same as with the desilverizing-kettle of Parkes process (§247).

The mode of operation in outline with the method by thirds is as follows: In the central kettle the lead bullion is melted down, drossed and, if necessary, poled. The fire below is then withdrawn and transferred to a neighboring kettle. The cooling is promoted by sprinkling water on the surface from a rose. Crusts adhering to the sides of the kettle are pushed down into the lead, where

[1] *Berg. Hüttenm. Z.*, 1863, XXII, 64, 69, 77.

[2] Stetefeldt, *Ibid.*, pp. 297, 381.

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they melt again. This is the work of one man, who also stirs the metal continuously until the smooth surface becomes rough with crystals. His partner now inserts at the rim of the kettle a long-handled suspended skimmer that has been warmed, and works it across the bottom of the kettle to the opposite side, then back to the middle, where, after jerking it to remove the liquid lead, he discharges the dry crystals into the neighboring kettle, generally the one to the right ("down the house"). The operation is continued until two-thirds of the contents of the kettle have been removed in the form of crystals. The liquid lead is then ladled into the kettle on the left ("up the house"). To the kettle at the right, being two-thirds full of crystals, one-third of lead of the same tenor is added, and the kettle at the left, being one-third full of liquid lead, is filled with a corresponding amount (two-thirds) of lead of its tenor. The kettles are heated, and the cooling, crystallizing, and ladling carried on in the same way as in the original bullion kettle. This becomes again filled from the crystals of the kettle on the left and the liquid lead of that on the right. Thus the operations are continued, the lead of the kettles to the right decreasing in tenor till that of the last one, the market-pot, assays from 0.3 to 0.5 oz. Ag per ton; that to the left increasing till the maximum of 500 oz. is reached.

From the foregoing it will be seen that before the whole plant can be in working order quite a number of crystallizations have to be carried on, so as to have on hand the necessary amounts of lead of different Ag contents required to fill the kettles.

Pattinson's process in its original form is still in use in England, Freiberg, Saxony, and perhaps some other places. As it is improbable that it will be introduced anywhere in the United States, this general outline will suffice. Full details are given in the works of Percy,[1] Kerl,[2] Schnabel-Louis,[3] Stölzel,[4] Roswag,[5] Grüner,[6] and especially the paper by Teichmann.[7]

In order to reduce the hard work necessary in withdrawing the crystals and ladling out the lead, as well as to insure a more regular crystallization and better separation of crystals and liquid lead, machinery was introduced into Pattinson's process; but the two main modifications of the original process, which are in use to-day, are those by Luce and Rozan, and by Tredinnick. In both the metal is stirred by steam, and the liquid lead drawn off, leaving the crystals in the kettle.

221. Luce and Rozan Process (Steam-Pattinson Process), General.[8]— The advantages of steam are, that it causes a regular crystallization and a good separation of the lead from the crystals, and that it poles the lead, which

[1] "Metallurgy of Lead," Murray, London, 1870, p. 121.
[2] "Grundriss der Metallhüttenkunde," Felix, Leipsic, 1881, p. 225.
[3] "Handbook of Metallurgy," Macmillan, London, 1905, Vol. I, p. 625.
[4] "Metallurgie," Vieweg, Brunswick, 1863–1886, p. 1122.
[5] "La désargentation de plomb," Dunod, Paris, 1884, pp. 211 and 267.
[6] *Ann. Min.*, 1868, XIII, p. 379.
[7] *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1867, XV, 40.
[8] Luce and Rozan, *Ann. Min.*, 1873, III, 160.
Cookson, *Eng. Min. J.*, 1881, XXX, 239.

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being much exposed to the influence of the air becomes purified. Thus moderately pure lead bullion, containing from 0.5 to 0.75 per cent. of foreign metals, can be desilverized without previous softening. It is claimed that lead with a little Sb and Cu is even preferable, as less dross forms than would be the case if it were free from these metals. Of course, lead containing appreciable amounts of As and Sb has to be softened with this process, as with any other, before it can be satisfactorily desilverized.

The way in which the process is carried out at Přibram,[1] Bohemia, may serve as an example. It was replaced about 1911 by the Parkes process.[2]

222. Luce and Rozan Process at Přibram, Bohemia.—The plant, shown in Figs. 467, 468, and 470, is seen to consist of two melting-pans, *a*, one crystallizing-pot, *l*, and two large conical molds. The steam crane is not shown; it is placed on the side of the crystallizing-pot, and serves to transfer the cakes of lead from the molds to the storage-place, and thence to the melting-pans, and to tip the latter. The trough-shaped cast-iron melting-pans, *a*, each holding 15,430 lb. of lead, are placed behind, and 2 ft. 4 in. above, the top of the crystallizer, *l*. They rest with their rims on cast-iron frame, *b*, and are emptied by tipping, by means of the crane, over inclined plate, *c*, which discharges the lead through stationary cast-iron trough, *d*, and a movable sheet-iron trough (not shown), into crystallizer, *l*. Each pan has its separate fireplace (Figs. 467 and 472) on the side, from which the gases, after passing upward (Fig. 467) through a long flue, surround the bottom of the pan, and descend either directly through flue *e* (Figs. 467 and 468) to the chimney, or first encircle the upper part of crystallizer *l* (Figs. 467 and 468), and then pass off through flue, *f* (Fig. 468); the passage of the gases is regulated by dampers, *g* (Fig. 467) and *h* (Fig. 468). On the oval hearth (Fig. 470) are built two small walls, *k* (Figs. 467, 468, 470), in order that the flame may pass close to the pan. Any lead coming from a leaking pan collects in the lowest part of the hearth (Fig. 467), tamped with brasque, and is discharged outside of the brickwork.

The crystallizer, *l* (Figs. 467 and 468), is a flat-bottomed cylindrical pot holding 44,100 lb. of lead, or nearly three times as much as one melting-pan. It has (Fig. 474) near the bottom two spouts, *A* and *B*, closed by slide-valves (Figs. 478, 479, 480), for discharging the lead. At a right angle to the plane of these spouts is steam-inlet, *C* (Figs. 473 and 474). The pot rests (Figs. 467 and 472) on cast-iron frame, *p*, supported by four cast-iron pillars, *o*. The top of the crystallizer (Fig. 467) is covered with a conical hood, ending in a sheet-iron pipe, through which steam and dust are carried off to be condensed. The hood has three openings closed by doors—one at the front above the steam-inlet, and two at the sides above the lead-spouts; and a small hole near the top, for the water inlet-pipe. The crystallizer is fired (Figs. 467 and 472) from the passage below melting-pan, *a*; the flames pass along the bottom, turn to the left, and encircle the lower part of the pot; they are checked by being forced

[1] Zdráhal, *Oest. Jahrb.*, 1890, XXXVIII, 1.
Private notes, 1890, and correspondence, 1897.

[2] Hahn, *Eng. Min. J.*, 1911, XCII, 1039.

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to make their way through the narrow passage, *q*, before passing downward and off through flue, *r*. On either side of the large central fireplace is (Figs. 471 and 472) a smaller one, which serves to heat discharge-spouts, *m*, before using them. Each of the discharge-spouts, *A* and *B* (Fig. 474), has a perforated cast-iron straining-plate to keep back the crystals when the liquid lead is being run off. These are held in place by wrought-iron arms, *b*, and cast-iron frame, *c* (Figs. 474 and 481), which is fastened by key-bolts to baffle-plate, *d*. The spouts are closed by a slide-valve (Figs. 478, 479, 480). To the flange of the spout (Fig. 476) is fastened with countersunk screws and a red-lead cement, a plate (Fig. 477) of the same form, with one face planed smooth, having four openings to correspond to those of the flange; *i.e.*, the central lead-discharge and the three holes near the rounded corners. Through these bolts, *m'* and *m*, are passed, *m'* serving as a pivot for lever, *o*, and *m* to tighten guide, *n*. To the lever is fastened plate, *p*, also having one planed face. In Fig. 478 is shown the position of the lever when the discharge is closed. In order to open it, the nuts of screw-bolts, *m*, are loosened, and the lever pushed into the second position, shown by the dotted lines. The lead from the crystallizer passes through the two lead spouts into two tapering molds (Figs. 470 and 482), each of which holds about 6610 lb. of lead. The steam-inlet consists of the following parts (Fig. 473). On the flat bottom of the crystallizer are four bosses, *f*, into which fit screws, *g*. On the collar of these is placed and keyed the cast-iron circular baffle-plate, *d*, with its small opening, *c*, in the center. It serves to distribute the steam evenly, and to make it rise regularly in the pot. From it is suspended by an eye-bolt, *i*, with hexagonal eye, nozzle, *h*, into which is screwed steam-pipe, *e*. Through it passes rod, *k*, moved to and fro at one end by the thread and cross-bar; the other end, which is conical, fits into the conical valve-seat of nozzle, *h*, and closes or opens the steam-outlet. The steam entering at *S* (Fig. 474) passes through the small annular space between pipe and rod, and out at *h*, when the valve is open.

The method of working is the one by thirds. The mode of conducting the process is simple. Suppose the process to be going on at the stage when the liquid lead has been drained off from the crystallizer; the valves have been again closed and the crystals liquefied. One pan will be full of liquid lead of the same tenor in Ag as the melted crystals to be discharged into the crystallizing pot, while the other will contain two cakes of lead that are being melted down. They will have the same Ag contents as the crystals remaining in the crystallizer after the operation to be described has taken place. The melting down of two cakes takes about 6 hr.

The lead from the pan is run out by inserting two hooks, fastened to the chain suspended from the pulley of the crane, into the ears of the pan, and raising it slowly. After the lead has been discharged into the crystallizer the doors of the hood are closed and steam introduced for 2 min. and shut off again; when the pulverulent dross which has collected on the surface is removed. Now the crystallization proper begins. The fire below the crystallizer is withdrawn and divided between the two small fireplaces on either side, from which the lead-

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spouts are warmed. Steam is turned on, and a small jet of water is allowed to play at short intervals upon the surface of the lead. Every time the water is let on there are small explosions, and as soon as they become too violent the water is shut off again, while the steam enters continuously. The steam has 45 lb. pressure to the square inch, and care must be taken to have it dry. About 15 min. after introducing the steam, the lead that has been splashed up on the upper edge of the pot, or on the hood, has to be removed. The steam is shut off, the doors in the hood are opened one after the other, and the solidified lead-crusts broken off with a chisel-pointed bar, and pushed back into the lead. This shutting off of steam to remove the lead is repeated at least twice. While the crystallization is going on, the two cakes of lead required to fill again the melting-pan, just emptied into the crystallizer, are hoisted from below with the crane, and deposited one on top of the other in the pan. The crystallization is finished when the normal amount of steam can no longer overcome the resistance offered by the crystals. The result is that the boiling ceases, and the surface of the crystalline mass of lead shows only a slow, wave-like motion. Two-thirds of the original lead have now been converted into crystals, the lead having assumed a mushy consistency. Water and steam are shut off, the slide-valves are opened, and the liquid lead is discharged into the molds, which takes from 8 to 10 min. In these have previously been placed iron hooks, by which the cakes of lead, when cold, may be lifted out. The whole process of crystallization lasts about 1 hr. While the lead is running off, the fire from the two small fireplaces is returned to the grate below the crystallizer, and urged in preparation for the next operation.

The melting and other work require 3 hr., so that one operation lasts 4 hr.

Eleven crystallizations are necessary to obtain market lead from liquated lead bullion averaging 146.12 oz. Ag to the ton. Table 110 shows the average assay-value in ounces per ton of the different leads produced during a whole year's work.

Six charges are run in 24 hr.; two men working as partners attend to the crystallization, all the handling of the lead being done by the crane-man and his helper. The products of the process are rich lead, desilverized lead, dross, and flue-dust. The rich lead is cupelled, the desilverized lead is refined in a reverberatory furnace and molded into market lead, the dross and flue-dust are worked with similar products from other parts of the works. The output of metal is shown by Table 111.

The material consumed for desilverizing 100 tons of lead bullion excluding the refining of the lead, is: Charcoal, 25 bushels; bituminous coal, 26.60 tons (for melting and desilverizing); bituminous coal, 8.23 tons (for raising steam).

Balling[1] gives the life of a melting pan as 40 days, of a crystallizer as 120 days, which seems very low.

It has already been stated that before the process can be carried on normally, a number of preliminary crystallizations have to take place to furnish the necessary intermediary products. On account of the possibility that the process

[1] "Metallhüttenkunde," Springer. Berlin, 1885, p. 295.

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will be combined with the Parkes process in this country because of the behavior of bismuth (see Tredinnick process, §224), full details of the manner of starting the plant at Přibram are given, although they involve much repetition.

The outline represents 11 *operations*, each requiring one more *charge* than the one preceding, before market lead can be produced, and gives the 12 grades of argentiferous lead, I–XII, the values of which are given in Table 112. The "quintals" used at Přibram have been retained, as changing them into pounds would have deprived the scheme, complicated as it is, of the necessary clearness. One quintal is equal to 100 kilograms and these equal 220.46 lb. avoirdupois.

The start is of course made with blast furnace lead bullion, B. F., XI, assaying 142.91 oz. per ton, and 180 quintals is the amount given below in Operation 1, which upon crystallization would furnish 60 quintals liquid lead, L. L., XII, with 262.49 oz. Ag per ton and 120 quintals crystallized lead, C. L., X, with 93.33 oz. Ag per ton. As the separation according to thirds is never accurate, these figures are not actually obtained, and as further, in melting, poling, etc., a considerable amount of dross is formed, the start is made with 190 quintals instead of with 180 quintals, and in the operations 130 quintals are charged instead of 120. Even with these larger amounts, the weight of the market lead obtained ranges only from 105 to 112 quintals.

TABLE 110.—AG-ASSAYS OF LEADS IN LUCE-ROZAN PROCESS, PŘIBRAM, BOHEMIA

Ounces Ag per Ton

Market lead	Kettle No.											
	I	II	III	IV	V	VI	VII	VIII	IX	X	XI Lead bullion	XII Rich lead
0.43	0.87	1.75	3.21	6.41	10.21	18.96	29.16	40.83	55.41	93.33	142.91	262.49

TABLE 111.—DISTRIBUTION OF PB AND AG IN PRODUCTS

Recovered in	From 100 pounds lead bullion charged	From 100 ounces silver charged	From 100 pounds lead charged
Rich lead	42.99	97.36	42.58
Desilverized lead	44.76	0.17	45.01
Scrap lead	3.02	0.61	3.03
Dross	9.94	1.45	8.04
Flue-dust	0.46	0.07	0.35
Loss		0.34	9.99
Total	101.17	100.00	100.00

OPERATION 1

Charge 1. Blast furnace lead bullion 180, B. F. XI; gives 60 L., L., XII, which go to the cupelling furnace (cupel) and 120 C. L., X, which are liquefied in the crystallizer, drawn off into the molds, and stacked on the floor (floor).

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TABLE 112.—PRELIMINARY OPERATIONS OF LUCE-ROZAN PROCESS, PŘIBRAM, BOHEMIA

Operation No.	Charge						Gives liquid lead			Gives crystallized lead			
	Charge No.	Quintals	Kind	Source		Total quintals	Quintals	Kind	Destination	Quintals	Kind	Destination	
				Op. No.	Ch. No.							Operation No.	Charge No.
1	1	180	xi	Bl.	F.	180	60	xii	Cupel	120	x	Floor	
2	1	180	xi	Bl.	F.	180	60	xii	Cupel	120	x	2	2
2	2	120	x	2	1	180	60	xi	Floor	120	ix	Floor	
2		60	x	1	1								
3	1	120	xi	Bl.	F.	180	60	xii	Cupel	120	x	3	2
3		60	xi	2	2								
3	2	120	x	3	1	180	60	xi	Floor	120	ix	3	3
3		60	x	1	1								
3	3	120	ix	3	2	180	60	x	Floor	120	viii	Floor	
3		60	ix	2	2								
4	1	120	xi	Bl.	F.	180	60	xii	Cupel	120	x	4	2
4		60	xi	3	2								
4	2	120	x	4	1	180	60	xi	Floor	120	ix	4	3
4		60	x	3	3								
4	3	120	ix	3	2	180	60	x	Floor	120	viii	4	4
4		60	ix	2	2								
4	4	120	viii	4	3	180	60	ix	Floor	120	vii	Floor	
4		60	viii	3	3								
5	1	120	xi	Bl.	F.	180	60	xii	Cupel	120	x	5	2
5		60	xi	4	4								
5	2	120	x	5	1	180	60	xi	Floor	120	ix	5	3
5		60	x	4	3								
5	3	120	ix	5	2	180	60	x	Floor	120	viii	5	4
5		60	ix	4	4								
5	4	120	viii	5	3	180	60	ix	Floor	120	vii	5	5
5		60	viii	3	3								
5	5	120	vii	5	4	180	60	viii	Floor	120	vi	Floor	
5		60	vii	4	4								
6	1	120	xi	Bl.	F.	180	60	xii	Cupel	120	x	6	2
6		60	xi	5	2								
6	2	120	x	6	1	180	60	xi	Floor	120	ix	6	3
6		60	x	5	3								
6	3	120	ix	6	2	180	60	x	Floor	120	viii	6	4
6		60	ix	5	4								
6	4	120	viii	6	3	180	60	ix	Floor	120	vii	6	5
6		60	viii	5	5								
6	5	120	vii	6	4	180	60	viii	Floor	120	vi	6	6
6		60	vii	4	4								
6	6	120	vi	6	5	180	60	vii	Floor	120	v	Floor	
6		60	vi	5	5								
7	1	120	xi	Bl.	F.	180	60	xii	Cupel	120	x	7	2
7		60	xi	6	2								
7	2	120	x	7	1	180	60	xi	Floor	120	ix	7	3
7		60	x	6	3								

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TABLE 112.—OPERATION OF LUCE-ROZAN PROCESS, PŘIBRAM, BOHEMIA.—(*Continued*)

Operation No.	Charge						Gives liquid lead			Gives crystallized lead			
	Charge No.	Quintals	Kind	Source		Total quintals	Quintals	Kind	Destination	Quintals	Kind	Destination	
				Op. No.	Ch. No.							Operation No.	Charge No.
7	3	120	ix	7	2	180	60	x	Floor	120	viii	7	4
7		60	ix	6	4								
7	4	120	viii	7	3	180	60	ix	Floor	120	vii	7	5
7		60	viii	6	5								
7	5	120	vii	7	4	180	60	viii	Floor	120	vi	7	6
7		60	vii	6	6								
7	6	120	vi	7	5	180	60	vii	Floor	120	v	7	7
7		60	vi	5	5								
7	7	120	v	7	6	180	60	vi	Floor	120	iv	Floor	
7		60	v	6	6								
8	1	120	xi	Bl.	F.	180	60	xii	Cupel	120	x	8	2
8		60	xi	7	2								
8	2	120	x	8	1	180	60	xi	Floor	120	ix	8	3
8		60	x	7	3								
8	3	120	ix	8	2	180	60	x	Floor	120	viii	8	4
8		60	ix	7	4								
8	4	120	viii	8	3	180	60	ix	Floor	120	vii	8	5
8		60	viii	7	5								
8	5	120	vii	8	4	180	60	viii	Floor	120	vi	8	6
8		60	vii	7	6								
8	6	120	vi	8	5	180	60	vii	Floor	120	v	8	7
8		60	vi	7	7								
8	7	120	v	8	6	180	60	vi	Floor	120	iv	8	8
8		60	v	6	6								
8	8	120	iv	8	7	180	60	v	Floor	120	iii	Floor	
8		60	iv	7	7								
9	1	120	xi	Bl	F.	180	60	xii	Cupel	120	x	9	2
9		60	xi	8	2								
9	2	120	x	9	1	180	60	xi	Floor	120	ix	9	3
9		60	x	8	3								
9	3	120	ix	9	2	180	60	x	Floor	120	viii	9	4
9		60	ix	8	4								
9	4	120	viii	9	3	180	60	ix	Floor	120	vii	9	5
9		60	viii	8	5								
9	5	120	vii	9	4	180	60	viii	Floor	120	vi	9	6
9		60	vii	8	6								
9	6	120	vi	9	5	180	60	vii	Floor	120	v	9	7
9		60	vi	8	7								
9	7	120	v	9	6	180	60	vi	Floor	120	iv	9	8
9		60	v	8	8								
9	8	120	iv	9	7	180	60	v	Floor	120	iii	9	9
9		60	iv	7	7								

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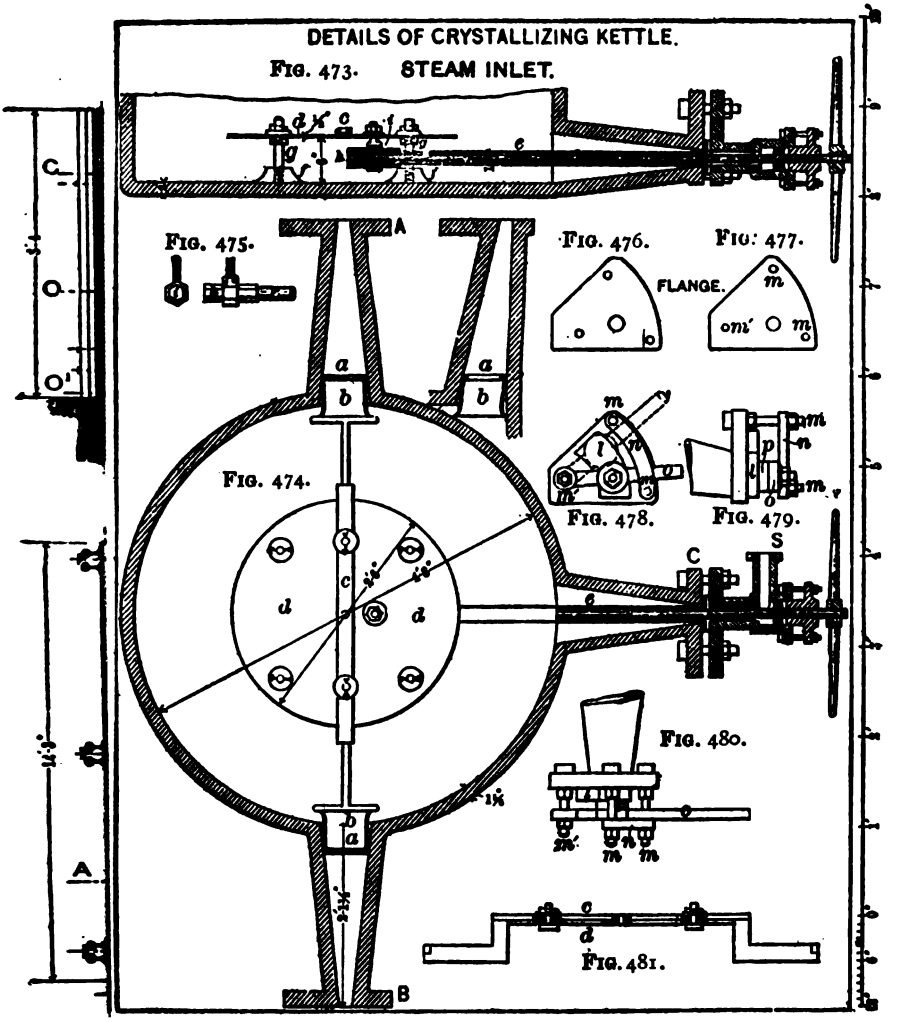

FIG. 482.

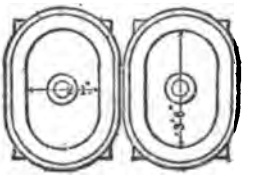

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TABLE 112.—OPERATION OF LUCE-ROZAN PROCESS, PŘÍBRAM, BOHEMIA.—(*Continued*)

Operation No.	Charge						Gives liquid lead			Gives crystallized lead			
				Source								Destination	
	Charge No.	Quintals	Kind	Op. No.	Ch. No.	Total quintals	Quintals	Kind	Destination	Quintals	Kind	Operation No.	Charge No.
9	9	120	iii	9	8	180	60	iv	Floor	120	ii	Floor	
9		60	iii	8	8								
10	1	120	xi	Bl.	F.	180	60	xii	Cupel	120	x	10	2
10		60	xi	9	2								
10	2	120	x	10	1	180	60	xi	Floor	120	ix	10	3
10		60	x	9	3								
10	3	120	ix	10	2	180	60	x	Floor	120	viii	10	4
10		60	ix	9	4								
10	4	120	viii	10	3	180	60	ix	Floor	120	vii	10	5
10		60	viii	9	5								
10	5	120	vii	10	4	180	60	viii	Floor	120	vi	10	6
10		60	vii	9	6								
10	6	120	vi	10	5	180	60	vii	Floor	120	v	10	7
10		60	vi	9	7								
10	7	120	v	10	6	180	60	vi	Floor	120	iv	10	8
10		60	v	9	8								
10	8	120	iv	10	7	180	60	v	Floor	120	iii	10	9
10		60	iv	9	9								
10	9	120	iii	10	8	180	60	iv	Floor	120	ii	10	10
10		60	iii	8	8								
10	10	120	ii	10	9	180	60	iii	Floor	120	i	Floor	
10		60	ii	9	9								
11	1	120	xi	Bl.	F.	180	60	xii	Cupel	120	x	11	2
11		60	xi	10	2								
11	2	120	x	11	1	180	60	xi	Floor	120	ix	11	3
11		60	x	10	3								
11	3	120	ix	11	2	180	60	x	Floor	120	viii	11	4
11		60	ix	10	4								
11	4	120	viii	11	3	180	60	ix	Floor	120	vii	11	5
11		60	viii	10	5								
11	5	120	vii	11	4	180	60	viii	Floor	120	vi	11	6
11		60	vii	10	6								
11	6	120	vi	11	5	180	60	vii	Floor	120	v	11	7
11		60	vi	10	7								
11	7	120	v	11	6	180	60	vi	Floor	120	iv	11	8
11		60	v	10	8								
11	8	120	iv	11	7	180	60	v	Floor	120	iii	11	9
11		60	iv	10	9								
11	9	120	iii	11	8	180	60	iv	Floor	120	ii	11	10
11		60	iii	10	10								
11	10	120	ii	11	9	180	60	iii	Floor	120	i	11	11
11		60	ii	9	9								
11	11	120	i	11	10	180	60	ii	Floor	120	x	Market lead[1]	
11		60	i	10	10								

[1] After it has been refined in a reverberatory furnace to remove the As and Sb it still retains.

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OPERATION II

Charge 1. 180 B. F., XI; gives 60 L. L., XII, to cupel, and 120 C. L., X, remaining in the crystallizing kettle; there are added from the melting pots 60 C. L., X, from Operation I, charge 1, to make up

Charge 2. 180, X; gives 60 L. L., XI, to floor, and 120 C. L., IX, to floor.

OPERATION III

Charge 1. 120 B. F., XI, and 60, XI, from Operation II, charge 2, gives 60 L. L., XII, to cupel, and 120 C. L., X; add remaining 60 C. L., X, from Operation I, charge 1, to make up.

Charge 2. 180, X; gives 60 L. L., XI, to floor, and 120 C. L., IX; add 60 C. L., IX, from Operation II, charge 2, to make up.

Charge 3. 180, IX gives 60 L. L., X, to floor and 120 C. L., VIII, to floor, etc.

The preceding table gives systematically arranged all the preliminary steps necessary to arrive at the point where the first market lead is produced.

It thus requires 66 crystallizations to obtain the intermediary products necessary for normal work. One-half of the crystallized lead from Operation X, charge 10, remains over to be worked off subsequently.

In comparing the processes of Luce-Rozan and Pattinson, Cookson[1] comes to the conclusion that the former is to be preferred by far, as the softening of the lead is not so imperative, the cost of labor only 20 per cent., and that of fuel 40 per cent., of the cost by Pattinson's process; and, lastly, it produces only 33 per cent. of the amount of drosses obtained by Pattinson. The drawback of the greater original cost, and the continued expense of repair and renewal, is more than made up by the advantages.

223. Luce and Rozan Process, Eureka, Nev.[2]—In 1874 the Richmond Consolidated Mining Co. introduced this process at Eureka;[3] work continued until 1889.[4] The capacity of the first crystallizer was 22 tons; it was increased in 1878 to 50 tons.[5] The lead bullion, averaging 100 oz. Ag and 1.5 oz. Au per ton, was desilverized by 4 men in 11 operations of 20 min. each to 0.5 oz., at a cost of $13.00 per ton. According to A. Wetzstein,[6] it was found that enlarging the crystallizer from a capacity of 22 to 50 tons permitted making 10 operations in 24 hr. instead of the usual 6 or 8, as the separation of crystals from liquid lead was quicker and better. Curtis[7] gives the following values, in ounces Ag per ton, for the products: 460 (rich lead), 105, 100, 75,

[1] *Eng. Min. J.*, 1879, XXVII, 258.

[2] Curtis, "Silver-lead Deposits of Eureka, Nev.," monograph U. S. Geol. Survey, 1884, 163.

[3] Ingalls, W. R., "Lead and Zinc in the United States," McGraw-Hill Book Co., New York, 1908, 77.

[4] Letter of R. K. Morrison, Supt., March 7, 1901.

[5] Letter of A. Tredinnick, May 15, 1903.

[6] "The Improved Pattinson Process," Butte, Mont., n. d.; *Min. Ind.*, 1902, XI, 451.

[7] *Loc. cit.*

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50, 30, 18, 9, 5, 2.5, 1.25 (market lead); Tredinnick[1] states that the Ag-content of the market lead never exceeded 0.5 oz. Ag per ton.

224. Tredinnick Process, General.[2]—This process, patented by Stephen Tredinnick, is in part a combination of the original Pattinson and Luce-Rozan processes. It has a crystallizer for each grade of lead as has the Pattinson; it stirs the lead with steam and drains off the liquid lead as does the Luce-Rozan. The innovation is that each crystallizer is mounted on a hydraulic platform so that it can be raised when it is to deliver lead, and lowered when it receives lead. A battery of usually 11 cylindrical cast-iron vessels is built in a row which has at one end a kettle for market lead, and at the other one for enriched lead which is to be cupelled or electrolyzed.

The process has not been used so far for desilverizing lead bullion, but was put into operation at Omaha, Nev., for the recovery of Bi from lead which had been desilverized by the Parkes process. It did this work satisfactorily, but has been abandoned, as there was not enough Bi-bearing lead in the market to warrant running the plant. At present, the small amount of bismuthic lead in the market is mixed with lead bullion made from Bi-bearing leady materials, and treated by the Betts process (290).

There is no question that the Tredinnick process will give excellent results with silver, as this metal is more satisfactorily concentrated in an enriched lead than is bismuth. Of course, it will not make corroding lead from Cu-bearing lead bullion, as Pattinsonizing does not eliminate the Cu to the required degree. This is shown in Table 113 given by Newnam,[3] who studied this feature.

TABLE 113.—DECOPPERIZATION OF LEAD BY PATTINSONIZING

Kettle	Cu, Per cent.
Bullion charged	0.084
Enriched lead	0.044
12	0.052
11	0.080
10	0.084
9	0.056
8	0.056
7	0.064
6	0.056
5	0.052
4	0.048
3	0.044
2	0.048
1	0.040
Impoverished lead	0.038

[1] *Loc. cit.*

[2] U. S. Patent, No. 662836, Nov. 27, 1900.
Austin, *Min. Sc. Press*, 1907, XCIV, 89.
Newnam, *Bull. A. I. M. E.*, May, 1917, 749.
Wetzstein, *Min. Ind.*, 1908, XVII, 605.

[3] *Loc. cit.*

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The following discussion, based on the above paper by Newnam quoted, deals with the Tredinnick process for the treatment of lead with 0.33 percent. Bi, desilverized by the Parkes process, with the object of obtaining lead with 1 per cent. Bi to be electrolyzed by the Betts process and corroding lead with 0.050 per cent. Bi to go to market.

The freezing-point curve of the alloy series Pb–Bi, Fig. 13, shows that Bi, melting at 268° C. and Pb. melting at 321.5° C., form an eutectic with 44 per cent. Pb, freezing at 125° C.; and solid solutions up to 10 per cent. Pb in Bi and 10 per cent. Bi in Pb. If we have molten lead with a small amount, say 0.33 per cent. Bi, and cool to 321.5° C. there will fall out of solution at first lead and then, with falling temperature, lead with increasing amounts of Bi, until this reaches its maximum of 10 per cent. just above the eutectic line at 125° C. The existence of this solid solution of Bi in Pb gives the reason for the difficulty of obtaining market lead with less than 0.05 per cent. Bi.

In crystallizing Ag-bearing lead with the method by thirds, the crystals are about half or 0.5 times as rich, and the liquid lead twice or 2 times as rich in Ag as the original lead bullion. With bismuth, the liquid lead contains about 1.4 times the percentage of Bi of the lead in the charging kettle, which represents approximately a 40-per cent. enrichment; the liquid lead contains about 0.8 times the Bi-content of the original lead, which is equal approximately to a 20-per cent. impoverishment. These facts are shown in Table 114.

TABLE 114.—CHANGES IN BI- AND AG-CONTENT BY PATTINSONIZING

Kettle No	1	2	3	4	5	6	7	8	9	10	11	
Bi per cent.....	0.050	0.063	0.081	0.104	0.130	0.164	0.206	0.260	0.330	0.440	0.580	0.770	1.000
Ag oz. per ton	¼	½	1	2	4	7	12	20	35	60	100	180	300

The missing 40 per cent., due to the formation of solid solutions, is the cause of the tendency of the Bi-contents of the kettles, between the corroding and the charging ends, to increase steadily in their tenors of Bi. This tendency has been termed the "crawl."

In a series of 11 crystallizing kettles, each having a working capacity of 63 tons lead, No. 1 represents the Bi-poor and No. 11 the Bi-rich end; No. 8 is charged with 63 tons of lead with 0.300 per cent. Bi. Crystallizing No. 8 will furnish

"No. 7 with 21 tons liquid carrying 0.240 per cent. Bi.

The kettles stand thus:

		No. 7	No. 8	No. 9
I.	Tons.....	63	Empty	21
	Per cent. Bi.....	0.240		0.420

Now crystallize No. 7, which will furnish:

No. 8 with 21 tons liquid carrying 0.336 per cent. Bi.

No. 6 with 42 tons crystals carrying 0.192 per cent. Bi.

Charge No. 8 with 42 tons of lead at 0.300 per cent. Bi; the average contents of No. 8 will then be 63 tons carrying 0.312 per cent. Bi.

Crystallize No. 8 which will furnish:

No. 9 with 21 tons liquid carrying 0.437 per cent. Bi.

No. 7 with 42 tons crystals carrying 0.250 per cent. Bi.

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Observe that the liquid from No. 7 to No. 8 is 0.336 per cent. Bi, or 12 per cent. higher than it should be. This before it has been subjected to the influence of any of the higher kettles. It is therefore correct to say that the liquid from No. 6 to No. 7 will increase 12 per cent. every time the operations proceed up the line. Hence, as No. 7 is approached the next time, No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.268 per cent. Bi.

The kettles will stand thus:

		No. 7	No. 8	No. 9
II.	Tons	63	Empty	42
	Per cent. Bi	0.256		0.428

Crystallize No. 7 which will furnish:

No. 8 with 21 tons liquid carrying 0.358 per cent. Bi.

Charging No. 8 kettle with 42 tons lead at 0.300 per cent. Bi; it will contain 63 tons of lead carrying 0.319 per cent. Bi.

Crystallize No. 8, which will furnish:

No. 9 with 21 tons liquid carrying 0.446 per cent. Bi.

No. 7 with 42 tons crystals carrying 0.225 per cent. Bi.

No. 9 now contains 63 tons of lead at 0.434 per cent. Bi, which crystallized will furnish:

No. 8 with 42 tons crystals carrying 0.347 per cent. Bi.

As No. 7 is approached the next time, No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.300 per cent. Bi.

The kettles will stand thus:

		No. 7	No. 8	No. 9
III.	Tons	63	42	Empty
	Per cent. Bi.	0.270	0.347	

Crystallize No. 7 which will furnish:

No. 8 with 21 tons liquid at 0.378 per cent. Bi. It will now contain 63 tons carrying 0.357 per cent. Bi.

Crystallize No. 8 which will furnish:

No. 9 with 21 tons liquid carrying 0.500 per cent. Bi.

No. 7 with 42 tons crystals carrying 0.286 per cent. Bi.

As No. 7 is again approached, No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.336 per cent. Bi.

The kettles will stand thus:

		No. 7	No. 8	No. 9
IV.	Tons	63	Empty	21
	Per cent. Bi	0.302		0.500

Crystallize No. 7 which will furnish:

No. 8 with 21 tons liquid carrying 0.422 per cent. Bi.

Charge No. 8 with 42 tons lead at 0.300 per cent. Bi, and it will contain 63 tons carrying 0.341 per cent. Bi.

Crystallize No. 8 which will furnish:

No. 9 with 21 tons liquid carrying 0.477 per cent. Bi.

No. 7 with 42 tons crystals carrying 0.273 per cent. Bi.

As No. 7 is again approached, No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.376 per cent. Bi.

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The kettles will stand:

		No. 7	No. 8	No. 9
V.	Tons	63	Empty	42
	Per cent. Bi	0.341		0.488

To Recapitulate

	No. 7	No. 8	No. 9
I	0.240	0.300	0.420
II	0.256	0.312	0.437
III	0.270	0.319	0.446
IV	0.302	0.357	0.500
V	0.341	0.341	0.477

It can readily be seen that this is an accumulative "crawl" and while it is aggravated each time No. 9 is brought into play and in like manner more so when No. 10 and No. 11 are drawn upon, yet it would exist were there no kettles above No. 8. Furthermore, the percentage rate of "crawl" of No. 2 or any other kettle will be the same as that of No. 7."

This crawl, or enrichment in Bi-content over that which is normal for a kettle, can be corrected either by tapping a kettle which is overgrade into the kettle of next higher grade, or by increasing the number of crystallizations. A combination of the two methods is effected by the so-called "jumps" in which the 21 tons of melted crystals from kettle No. 1 are tapped into kettle No. 2 holding 42 tons of crystals, and then working straight up the line without tapping back any crystals. Thus the crystals receive an extra operation, and one-third liquid lead is pushed up the line by successive stages. Tables 115 and 116 show the increase in Bi-content without and with jumping (J) in a 10-kettle plant.

TABLE 115.—INCREASE IN BI-CONTENT WITHOUT "JUMPING"

Corroding "C"	1	2	3	4	5	6	7	8	9	10	Anode "A"
0.061	0.075	0.110	0.170	0.208	0.251	0.315	0.400	0.468	0.675		
0.061	0.073	0.131	0.161	0.211	0.285	0.348	0.401	0.529			
0.067	0.079	0.129	0.169	0.229	0.295	0.358	0.412	0.462			
0.071	0.093	0.138	0.148	0.233	0.276	0.346	0.454	0.498	0.793	0.862	0.878
0.081	0.093	0.138	0.181	0.237	0.287	0.358	0.407	0.562	0.641		
0.076	0.093	0.154	0.197	0.254	0.299	0.331	0.399	0.529			
0.076	0.095	0.148	0.197	0.249	0.299	0.362	0.445	0.482			
0.078	0.102	0.142	0.199	0.257	0.328	0.388	0.486	0.476	0.691		
0.074	0.092	0.146	0.204	0.261	0.345	0.366	0.413	0.548			
0.086	0.104	0.158	0.202	0.262	0.316	0.370	0.463	0.495			
0.088	0.117	0.164	0.219	0.274	0.342	0.405	0.445	0.478			
0.095	0.116	0.168	0.225	0.280	0.353	0.408	0.423	0.571	0.796		

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TABLE 116.—INCREASE IN BI-CONTENT WITH "JUMPING"

Corroding "C"	1	2	3	4	5	6	7	8	9	10	Anode "A"
								C			
0.050	0.063	0.079	0.103	0.136	0.187	0.236	0.316	0.390			
0.048	0.060	0.086	0.112	0.157	0.202	0.265	0.378	0.442	0.521		
0.048	0.055	0.096	0.127	0.180	0.219	0.296	0.367	0.425			
0.045	0.058	0.099	0.136	0.177	0.227	0.285	0.357	0.428			
0.056	0.069	0.109	0.141	0.192	0.248	0.307	0.373	0.422	0.581	0.775	1.045
	0.073	0.110	0.153	0.203	0.254	0.314	0.371	0.475	0.636		
	J	0.081	0.119	0.161	0.212	0.282	0.325	0.409			
0.048	0.058	0.097	0.132	0.174	0.236	0.291	0.359	0.436			
0.049	0.064	0.102	0.137	0.179	0.237	0.294	0.364	0.415			
0.058	0.074	0.109	0.147	0.197	0.245	0.296	0.369	0.417	0.588	0.829	1.103
	0.083	0.111	0.150	0.190	0.253	0.297	0.358	0.436	0.631		
	J	0.080	0.114	0.158	0.218	0.274	0.318	0.373	0.500		
0.053	0.070	0.103	0.136	0.178	0.234	0.282	0.345	0.423			
0.054	0.081	0.105	0.134	0.183	0.229	0.270	0.330	0.353			
	0.066	0.099	0.148	0.197	0.250	0.308	0.329	0.338	0.560	0.830	1.177
	J	0.079	0.124	0.163	0.204	0.275	0.294	0.325	0.430	0.595	0.853
0.046	0.062	0.094									
0.055	0.071										
0.056	0.076										

225. Plant.—The plant under consideration was constructed to treat daily about 200 tons lead with the method by thirds, and to produce from lead with 0.33 per cent. Bi, a corroding lead with 0.05 per cent. Bi, and an enriched lead with 1 per cent. Bi to be electrolyzed. This required 11 crystallizers, each holding 63 tons of lead. It consisted of one 180-ton storage reverberatory furnace for the lead to be treated; 11 crystallizers (Figs. 483–485), with a working capacity of 63 tons each, placed in a line at 12-ft. centers; one 150-ton molding furnace for corroding lead; one 42-ton kettle for molding anodes; one 150-ton reverberatory furnace for the reduction of drosses; one 18-ton ladle for charging the crystallizers, usually No. 8; one oblong spout kettle, filled with hot lead, for heating the tapping-spouts, which are kept in it until needed; one 25-ton electric traveling crane for handling the 18-ton ladle, for replacing defective kettles (done in 20 min.), and for handling in general.

226. Crystallizer.—Details of the crystallizer are shown in Figs. 483–485, which represent an elevation of a kettle-stand with connections and details of the kettle. A kettle, 7 ft. 1 in. in diameter and 6 ft. 3 in. deep, is surrounded and carried by a combustion chamber inclosed in a brick-lined sheet-steel casing. This is erected on an I-beam base with support-chairs, and is raised and lowered by a hydraulic ram, 26 in. in diameter, which has a stroke of 7 ft., and works under a hydraulic pressure of 500 lb. per sq. in. Steam is admitted on the side, close to the bottom, through a valve which delivers at the center under a heavy perforated plate, 45 in. in diameter. It enters the valve through a telescope pipe-connection at a pressure of 110 lb. per sq. in. (=173° C.). The

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cover of the kettle is a sheet-metal hood with four working-doors in the sides and a 13-in. opening in the top connected by a telescope pipe with a sheet-metal flue. The flue is provided with an 8-ft. exhaust fan, running at 300 r.p.m., which removes waste steam and powdery oxides of lead. Four inches above the top of the cover is a ¾-in. circular water-pipe, 24 in. in diameter, which has eight 3/32-in. holes on the lower side for the delivery of cooling water into the kettle

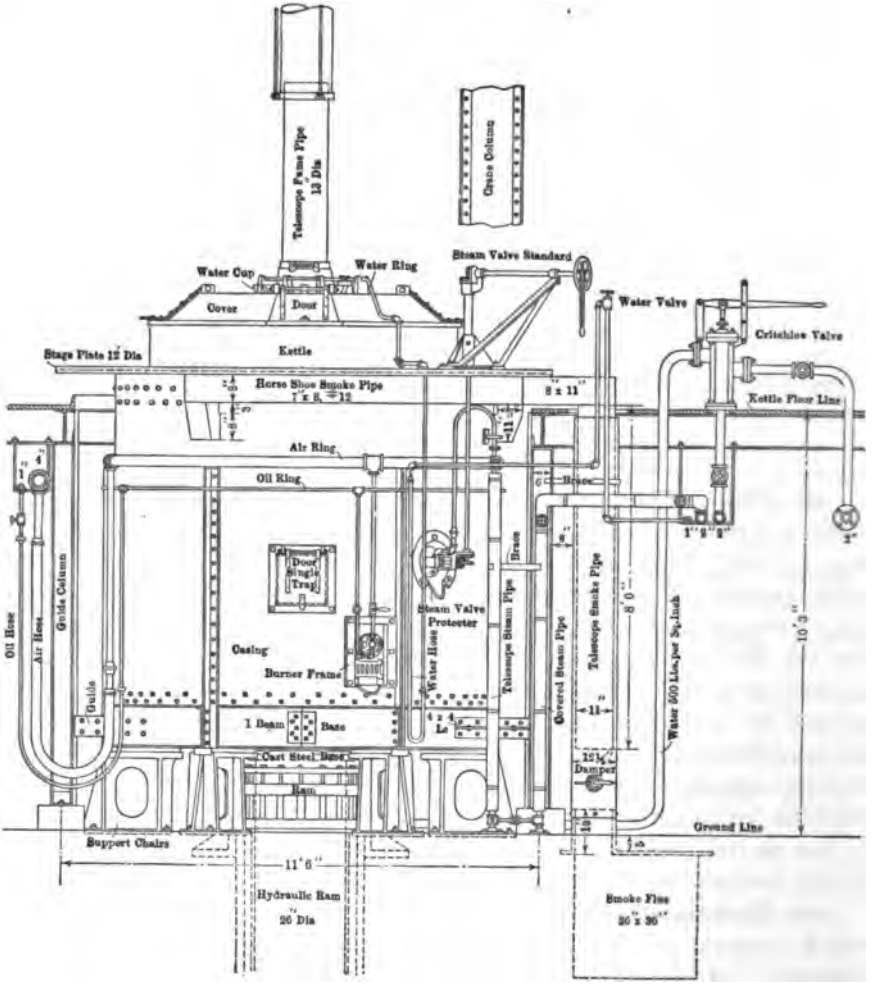

FIG. 483.—Tredinnick-Newnam crystallizer, elevation.

through funnel-shaped cups. The water-connection is made by a ¾-in. hose, 9 ft. long. Both water and steam are controlled from the second, or kettle, floor. A kettle is fired with oil through three atomizer burners operated with an oil-pressure of 40, and an air-pressure of 24 oz. per sq. in. Connection with the oil-tank is made by a ½-in. metallic hose, and with the air-receiver by a 2-in. flexible suction hose. The waste gases are withdrawn through three

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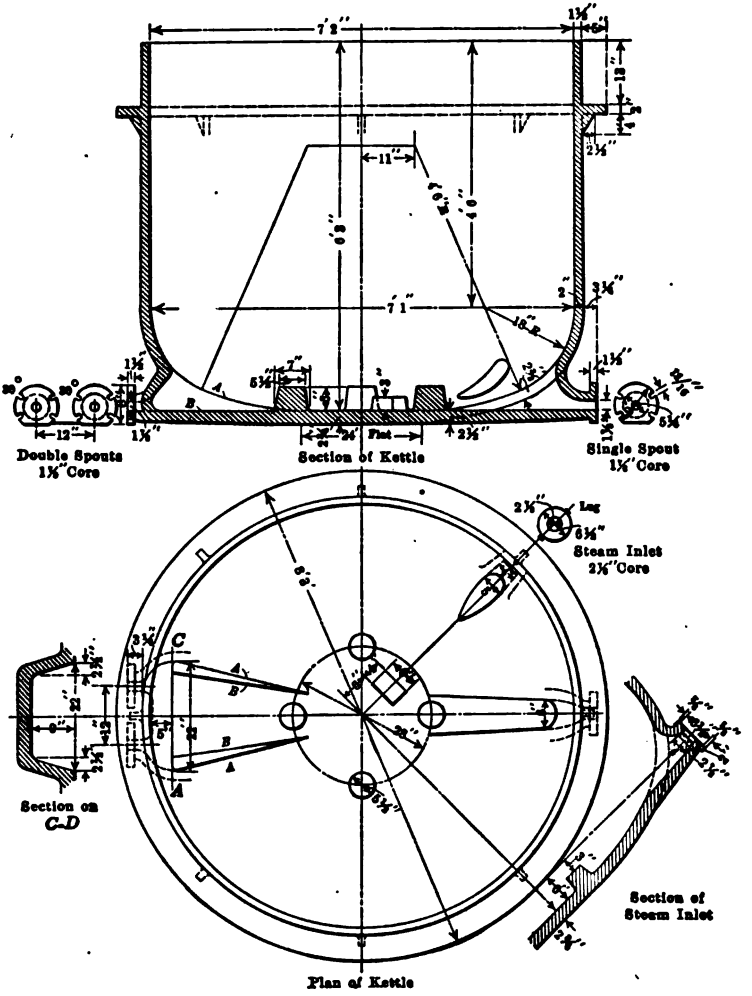
ports by a horse-shoe smoke-pipe connected by a telescope pipe with an underground smoke-flue provided with a 48-in. fan, running at 440 r.p.m.

The speed of operation is great in that it takes 45 min. to melt 45 tons of crystals, and 30 min. to crystallize and tap a charge of 63 tons.

227. Schedule of Operation (Newnam System).—In carrying out the process, two systems were tried, the Tredinnick[1] and the Newnam; the latter was



FIGS. 484 and 485.—Tredinnick-Newnam crystallizer, vertical section and plan.

adopted and is given here. The schedule of operations is shown in Fig. 486, a legend explaining the conventional signs used is added. The schedule represents an 11-kettle plant as it stands at the beginning of a double cycle.[2]

[1] *Op. cit.*, p. 754.

[2] A single cycle is one in which one tap of anode lead is made from kettle No. 11; a double cycle one in which two such taps are made. When kettles Nos. 1–8 are being operated, the work is termed the *Short String;* when kettles Nos. 9–11 come into play, the work is termed the *Long String.*

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"No. 1 kettle is full (63 tons) of melted crystals ready to crystallize. Kettles Nos. 2 to 10 inclusive are each two-thirds full (42 tons) of melted crystals, and No. 11 kettle is empty. The percentage of bismuth in each kettle is indicated below the kettles

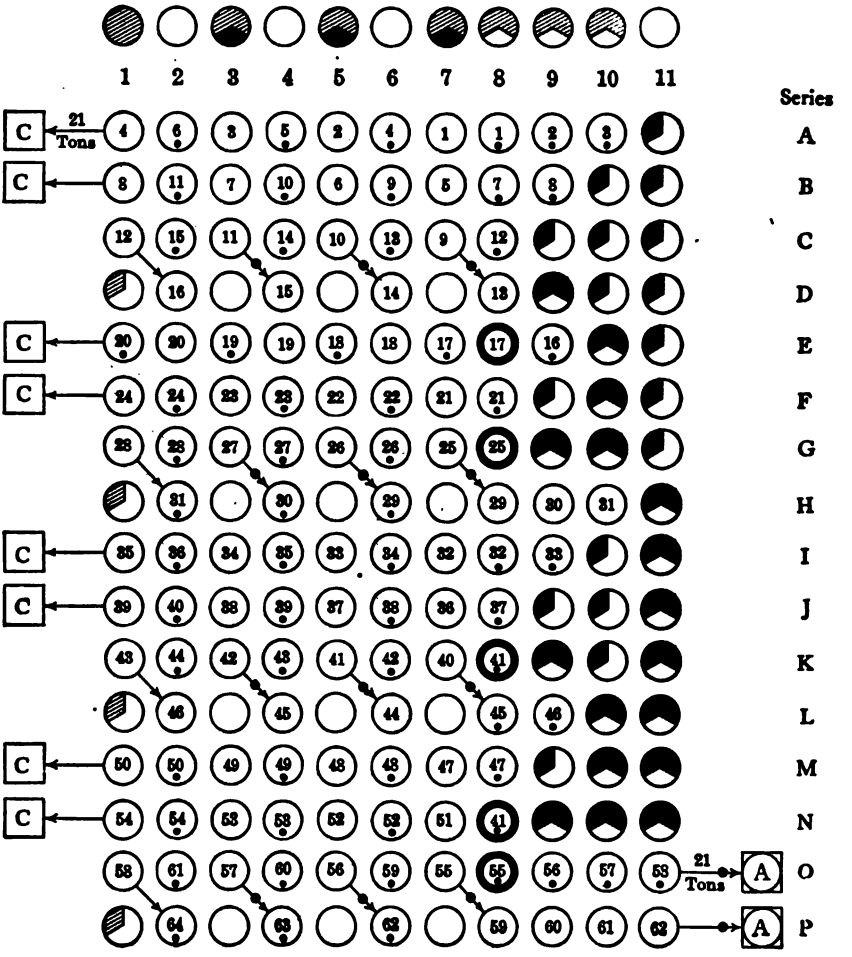

FIG. 486.—Newnam system for Tredinnick process.

and represents the approximate grade at which they should be maintained. All charges to be placed in No. 8 kettle.

The plant stands as follows:

Kettle No.	1	2	3	4	5	6	7	8	9	10	11	
	C											A
Per cent. Bi	0.060	0.063	0.081	0.104	0.130	0.164	0.206	0.260	0.330	0.440	0.580	0.770	1.000

No. 1 kettle is now crystallized, the kettle elevated and 21 tons of liquid are tapped into No. 2 kettle, making it a full kettle. The 42 tons of crystals in No. 1 kettle are

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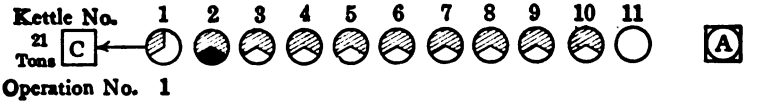
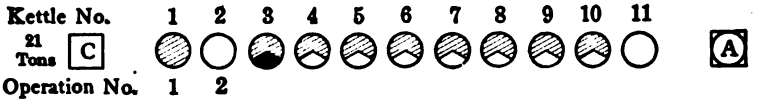
melted and 21 tons are tapped to the corroding lead-molding furnace, thus completing operation No. 1.

The plant stands as follows:



No. 2 kettle being full, it is now crystallized and 21 tons of liquid tapped into No. 3 kettle, filling it. As soon as the 42 tons of crystals in No. 2 are melted, the kettle is again elevated and contents tapped to No. 1 kettle, filling the latter and completing operation No. 2.

The plant stands as follows:



Kettles Nos. 1 and 3 now being full, it is evident that these two kettles may be operated at the same time, requiring two crews. As the first crew progresses up the string, the second crew would follow behind and as closely up as melting of the crystals would permit. In actual practice, usually two or three kettles intervene between the two crews. In order to avoid confusion in the cycle diagram, the work of the two crews is shown in alternating lines; thus, the first crew works out the string marked Series A,

LEGEND FOR FIG. 486.

Empty kettle.

Kettle containing 21 tons melted crystals.

Kettle containing 42 tons melted crystals.

Kettle containing 21 tons enriched liquid.

Kettle containing 42 tons enriched liquid.

Full kettle made up of 42 tons of melted crystals and 21 tons of enriched liquid.

Kettle containing charge of 42 tons.

Showing direction of the movement of melted crystals.

Showing direction of the movement of the enriched liquid.

C Corroding lead-molding furnace.

A Anode or enriched lead-molding furnace.

1 First crew.

1 Second crew.

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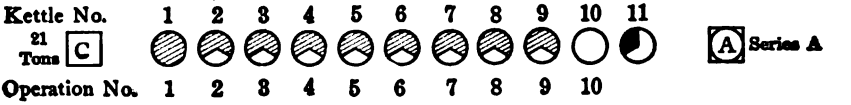
after which the second crew works out the Series B, although, as stated, in actual practice they follow as closely behind one another as plant conditions will permit.

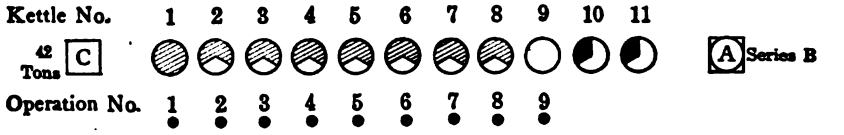
Returning to the last diagram, the first crew will successively crystallize the kettles 3 to 10 inclusive, and the operation on No. 10 will place 21 tons of liquid into the empty kettle No. 11. In each case the melted crystals are tapped toward the corroding end.

The plant will stand as follows:

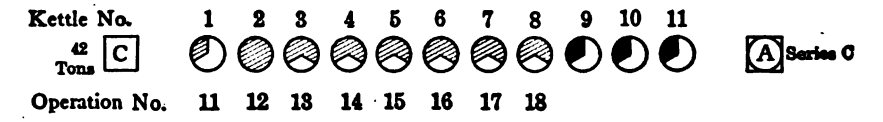


The first crew has thus made 10 operations on the long string, Series A. The second crew now begins on No. 1 kettle and, in like manner, successively operates the kettles Nos. 1 to 9 inclusive, and the operation on No 9 will place 21 tons of liquid into the empty kettle No. 10.

The plant will stand as follows:

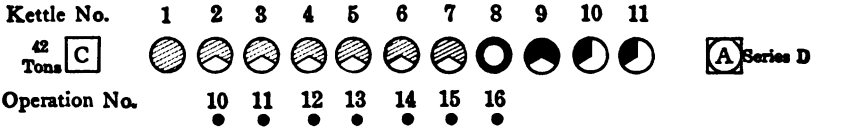


The second crew has thus made nine operations on the long string, Series B, and the first crew now returns to No. 1 kettle to work out its second string, or Series C. Owing to the necessity of correcting the grade of the kettles, as previously explained, a variation of the procedure now takes place in Series C. During this series none of the melted crystals are tapped toward the corroding end but are left in their original kettles with the exception of the crystals in No. 1, 21 tons of which, instead of being tapped to the corroding furnace, are tapped into No. 2 kettle, and the plant stands as follows:



The second crew returning to the corroding end to work out its short string, Series D, is obliged to pass over kettle No. 1, and begin operations on the full kettle No. 2, thus proceeding straight up the line with the operation on No. 8 kettle, placing a second tap of 21 tons of liquid in No. 9 kettle. As all crystals on this series are tapped back as usual, and as this will leave the charge kettle No. 8 empty, it is in order to give it a 42-ton charge.

The plant stands as follows:



No. 8 having been charged, the Series E will now be run out by the first crew, and it will be observed that the operation of No. 8 will fill kettle No. 9, which must then

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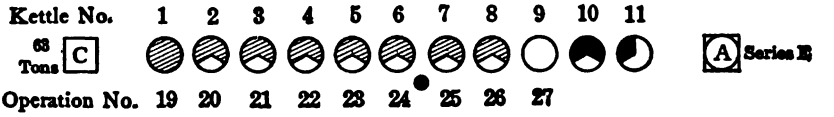
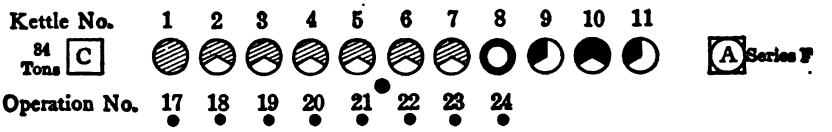
be worked, placing an additional 21 tons of liquid in No. 10, or 42 tons in all. Therefore, at the close of Series E by the first crew, the plant will stand as follows:

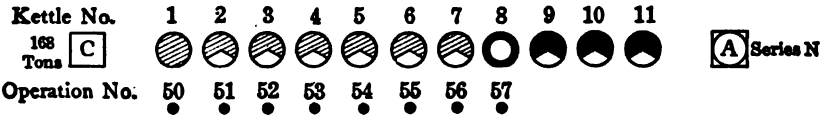


The second crew now begins its seventeenth consecutive operation with kettle No. 1 of the Series F, at the conclusion of which the plant will stand as follows:

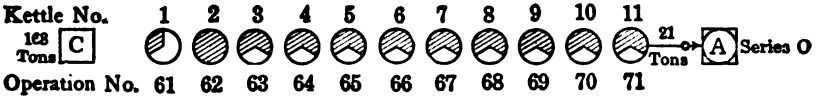


As No. 9 was not operated in Series F, there were no return crystals to No. 8; therefore, the second charge of 42 tons was introduced into that kettle.

On series G the correction for grade is again made by omitting the corroding-tap, and running 21 tons of No. 1 crystals into No. 2 kettle. The plant is steadily worked according to the above plan until the end of Series N, when the plant stands as follows:



As the ninth, tenth and eleventh kettles now contain 42 tons each, it is evident that the operations on Series O will include all kettles from 1 to 11 inclusive, and that 21 tons of anode lead will be tapped from No. 11 kettle into the anode molding kettle. As the correction of grade occurs at this time, none of the crystals produced on Series O will be tapped back, and at its termination the plant stands as follows:



The first crew will now run out the last string of the cycle, Series P, operating all the kettles from No. 2 to 11 inclusive and placing a second tap of 21 tons of anode lead into the anode kettle. All the crystals having been tapped down the line, at the termination of this series the plant returns to its original condition, as first shown, and a new cycle begins.

During this cycle the following operations were made:

First crew	71
Second crew	67
Total	138

As four corrections for grade were made during this period, $1\frac{3}{4}\frac{8}{}$ or $34\frac{1}{2}$ represents the number of operations made for each correction in grade. On the cycle shown, corroding lead was made in sets of two consecutive taps between corrections; now

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should these taps be made in groups of one, three, or four, an entirely different cycle would result, necessitating a separate diagram for each instance.

During the period, the ingoing and outgoing lead was as follows:

	Tons
Corroding lead, eight taps of 21 tons each	168
Anode lead, two taps of 21 tons each	42
Charge lead, five charges of 42 tons each	210

Dividing these tonnages by the total number of operations, 138, the tons of lead charged and produced may be expressed as follows: Tons per operation—charge 1.522, corroding 1.218, anode 0.304."

228. Details of An Operation.—The details of an operation are as follows:

"A full kettle (63 tons), skimmed clean and ready to crystallize, should have a temperature not much above the melting-point of lead, but the brickwork surrounding the kettle should be hot enough to prevent a crust from forming on the inside of the kettle. Correct temperature is an important factor, as lead too hot greatly prolongs the time of crystallizing, and produces an excessive amount of dross. In practice, the lead is tested by thrusting a broom handle into the molten metal; if the lead freezes to the handle the temperature is low enough to proceed with the operation. If the lead does not freeze to the handle, the kettle must be cooled with water, causing crusts to form which are pushed under the surface with a pole until the proper temperature is secured. As this also consumes time, it is necessary to keep a sharp watch on the temperature.

The kettle being in the proper condition, the operator opens the steam valve slowly until the surface of the lead is violently agitated. (In order to prevent slopping, the kettles are filled to within 15 in. of the top only.) Water is now cautiously introduced through the eight water cups in the cover. The operator regulates the admission of steam and water so that a maximum amount of water is introduced without causing explosions, the formation of chunks, or the slopping of lead through the cover doors.

The water cups occasionally become clogged with lead and have to be freed by a special punching rod in the hands of the barman.

Lead soon freezes to a crust on the upper ring of the kettle and to the cover. Periodically the water is turned off, the cover doors are thrown back, and the crusts barred down with a 6-ft. steel bar, 1 in. in diameter, having a chisel point. Considerable judgment must be exercised not to allow the crusts to become too thick, as in this case it is difficult to break them up with the steam, and they tend to form chunks. Also, too frequent barring down consumes time, as the water has to be turned off during that period.

In order to facilitate barring down, the inside of the cover and the upper ring of the kettle should be as smooth as it is possible to make them.

Soon crystals of lead, from $\frac{1}{16}$ to $\frac{1}{8}$ in. in diameter, begin to appear in the bath, and from this point on they multiply with ever increasing rapidity, the violent agitation by the steam keeping the crystals from adhering to one another.

When the consistency of two-thirds crystals has been reached, the surface of the lead appears as an exceedingly thick mass of boiling crystals. At this stage the water is shut off, the kettle barred down for the last time, and the crusts broken up by steam. Steam is now turned off, the kettle elevated and two hot spouts, just out of the spout kettle, are placed on the double lead-cocks. These cocks (which are kept hot with

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charcoal) are opened slowly, and the one-third of the charge is tapped to the adjoining kettle through a screen in the bottom of the kettle covering the double taps. This screen has 96 holes 3/16 in. diameter, 2-in. centers.[1]

As soon as the last crust has been barred down, the burners are started, and by the time the liquid has run out the temperature is rapidly rising.[2]

Under the proper conditions, a kettle can be crystallized in 15 min. An average operation, crystallizing and tapping, requires about 30 min.

It will be noted from the cycle diagram that any delay on one kettle affects the whole plant. Quick crystallizing and quick melting are therefore necessary to speed, and the more rapidly these operations are performed, the fewer kettles need intervene between the kettles that are being operated. Thus more kettles can be in operation at the same time.

On the short string, two crews are operating simultaneously, whereas on the long string as many as four crews may be working.

As soon as the crystals are melted, the lead is skimmed and tapped to the opposite adjacent kettle through the single lead-cock without a screen. The average melting period is about 45 min. This completes an "operation."

The kettles near the corroding end are much more difficult to operate than those near the anode end; also, the finished lead at the anode end appears much thicker than that at the corroding end. This is due to the crystals near the anode end being larger than those at the corroding end of the plant.

229. Dross, Fuel, Labor, Cost.—*Dross* is one of the most objectionable features of a Pattinson plant. In the early stages of the Tredinnick plant, 21 per cent. of the lead charged was skimmed off as dross. This was not only expensive to handle, reduce, and recharge, but it also left the lower kettles short of lead, and thereby reduced the tonnage materially.

It was found that throwing small quantities of fuel oil into the kettle on top of the crystals during the melting period, hastened the melting and reduced the dross formed during the crystallizing.

Handled in this manner, the reducing reverberatory furnace needed to be operated only 4 or 5 days a month, and the lead removed as dross amounted to not more than 2 to 3 per cent. of the lead charged.

During the crystallizing, about 50 lb. of litharge, fine as flour, is formed, which is drawn off by the telescope fume-pipe, and caught in the flue. A very strong draft is necessary to keep this fume from passing through the cover doors and affecting the health of the men.

The *fuel* requirements of the plant depend upon the number of operations. An average of 50 gal. of fuel oil is required per operation. Of this, 40 gal. pass through the burners and 10 gal. are thrown on top of the melting crystals in the kettle.

Labor.—The accompanying labor table is based on a crew sufficiently large to make from 50 to 70 operations each 24 hr."

[1] For some time the Hulst hydraulic press (U. S. Patent, No. 965464, July 26, 1910; *Eng. Min. J.*, 1910, XC, 853; and U. S. Patent, No. 1001525, Aug. 22, 1911; *Tr. A. I. M. E.*, 1914, XLIX, 537) was in operation in order to hasten the flow of liquid lead, and to obtain cleaner crystals.

[2] It is surprising how accurately a good crystallizer can judge the proportion of liquid and crystals in the finished kettle. A good man will seldom be in error over 2 tons, and this inequality may be eliminated on a subsequent operation by slightly over- or under-crystallizing as the case may require.

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TABLE 117.—LABOR

No. men	Rate	Duties	Per month
Day shift (10–12 hr.)			
1	$2.65	General foreman	$80.00
1	2.40	Burnerman	72.00
1	2.20	Burnerman assistant	66.00
2	2.25	Crystallizers	135.00
2	1.85	Barmen	111.00
1	1.75	Barman in training	52.50
1	2.00	Tapping and cleaning kettles	60.00
1	2.25	Crane-man	67.50
2	1.75	Clean-up and general utility	105.00
12			
Lap shift[1]			
1	$2.40	In charge at lap and noon time	$72.00
1	2.00	Crystallizer in training	60.00
1	1.85	Barman	55.50
2	1.75	Barmen in training	105.00
5			$1,041.50
Night shift		Same as above	$1,041.50
Reducing furnace			
1	$2.00	Furnace-man, 5 days per month	$10.00
1	1.65	Furnace-man helper, 5 days per month	8.25
3	1.65	Drossmen, 5 days per month	24.75
1		Day superintendent	125.00
1		Night superintendent	87.00
		Total per month	$2,338.00

[1] Five men coming on at 9, and working until 7 o'clock.

Cost.—An 11-kettle plant, as described, treating from 150 to 200 tons lead in 24 hr., costs about $85,000. The cost of an operation is about $7.50, which figure includes all overhead expenses, such as insurance, taxes, amortization, royalty, general expense. In comparison with the Parkes process (§260), the average cost per ton lead bullion charged in a moderate-size plant treating lead bullion with 100 oz. Ag per ton, with zinc at a normal figure of 5 cts. per lb., is about $4 per ton.

230. Thum Process.[1]—In this process, which has not yet been carried out in practice, the lead contents of stationary kettles are to be transferred from one to another by means of compressed air through rising pipes heated by electric resistance coils wound around them.

PARKES PROCESS

231. Parkes Process, General.—Parkes' process is based on the fact that if from 1 to 2 per cent. Zn is stirred into melted lead bullion, it will deprive the latter of its Ag, and form an alloy, which, being less fusible than lead, and having a lower specific gravity, will become hard and float on the surface of the lead, whence it can be removed and treated separately; the lead, which

[1] U. S. Patent No. 1106480, Aug. 10, 1904.

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has taken up some Zn, is refined, and is then ready for the market. Karsten[1] discovered in 1842 that argentiferous lead could be desilverized by the use of Zn, but his discovery could be applied in practice only when A. Parkes[2] found the means (1850–52) of working the zinc-silver-lead crust and refining the zincky lead.

In the United States[3] E. Balbach patented in 1864 a zinc desilverizing process in which lead bullion was made to flow into a kettle holding molten spelter. The contents were stirred; the crust was skimmed and cast into molds; the bars were liquated and the liquated crust distilled; and the zincky lead refined in a reverberatory furnace. The process was modified in time so that it became, about 1872, identical with that of Parkes. In 1873 there were in operation 14 establishments which refined lead bullion. The leading plants desilverizing lead bullion by means of the Parkes process in 1916 are given in Table 118.[4]

TABLE 118.—LEADING PARKES DESILVERIZING PLANTS OF THE UNITED STATES IN 1916

State	City	Company
California	Selby	Selby Smelting and Lead Co.
Idaho	Kellogg	Bunker Hill & Sullivan Smelting & Concentrating Co.
Illinois	Collinsville	St. Louis Smelting and Refining Co.
Illinois	Federal	Federal Lead Co.
Indiana	East Chicago	International Lead Refining Co.
Nebraska	Omaha	Omaha & Grant Smelting Co.
New Jersey	Newark	Balbach Smelting and Refining Co.
New Jersey	Maurer	American Smelting and Refining Co.

232. Behavior of Metals in Zinc Desilverization.—In zinc desilverization there have to be considered the behaviors of Ag, Au, Cu, Pt, Pd, Te, Bi, As, Sb, Sn, Ni, and Co likely to be present in the lead bullion, and of Cd and Fe in the spelter. Of the metals found in lead bullion, Ag, Au, Cu, Pt, Pd, and Te readily enter the zinc crust. The other metals do this only to a very small extent; they interfere, however, more or less with the work that the Zn is intended to do.

1. *Silver.*—Desilverization used to be explained by the statement that Ag had a greater affinity for Zn than for Pb, and therefore combined with the Zn when added to molten argentiferous lead. Roesing[5] modified this statement by saying that, though Ag had a greater affinity for Zn than for Pb, it had less affinity for Zn-bearing Pb than either for Zn or for pure Pb, and that this was the main reason why argentiferous lead could be desilverized by means of Zn.

[1] *Archiv.*, 1853, XXV, 196.
[2] Percy, "Lead," p. 148.
[3] Eurich, *Tr. A. I. M. E.*, 1912, XLIV, 741.
[4] Siebenthal, "Production of Lead in 1916," U. S. Geol. Survey.
[5] *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1899, XXXVII, 76.

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The crust which forms in zinc desilverization contains from 30 to 50 per cent. Pb; hence the theory has been put forth that ternary alloys of Zn, Ag, and Pb were formed. In practice it is common to speak of the "zinc-silver-lead crust." Kremann and Hofmeier[1] investigated the possible ternary alloys and came to the conclusion that in zinc desilverization no ternary alloys were formed, that the formation of the compound Ag_2Zn_5 caused desilverization; that Ag_2Zn_5 formed solid solutions with Zn, but not with Pb; and that the Pb in the crust was a mechanical admixture. They based their research in part upon the investigation of Petrenko,[2] whose freezing-point curve showed four chemical compounds of Zn and Ag, viz.: Ag_2Zn_5 with 60 per cent. Zn; Ag_2Zn_3 with 48 per cent.; AgZn with 37.7 per cent.; and Ag_3Zn_2 with 28.18 per cent. Zn.

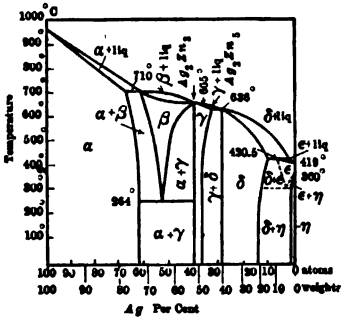

FIG. 487.—Alloy series Zn–Cu.

The curve of Petrenko has been superseded by the constitutional diagram of Carpenter and Whiteley[3] given in Fig. 487; Petrenko apparently having accepted the correction.[4] Fig. 487 shows six constituents: A solid solution α with from 0 to about 27 per cent. Zn, stable at ordinary temperature; a β-solution extending from about 20 to about 48 per cent. Zn, stable only above 264° C., below which it undergoes an eutectic transformation into $\alpha + \gamma$; a chemical compound Ag_2Zn_3 with 48 per cent. Zn, freezing at 665° C.; possibly a second chemical compound Ag_2Zn_5 with 60 per cent. Zn, freezing at 636° C., which forms solid solutions on the zinc side; a constituent ϵ stable only at high temperatures, which splits into δ and η at lower temperatures; solid solutions of Zn with not more than 1 per cent. Ag. Zinc desilverization therefore finds its explanation in the formation of the chemical compound Ag_2Zn_3 with 48 per cent. Zn, freezing at 665° C. The Pb present in the crust is a mechanical admixture of Zn-bearing Pb which owing to superficial oxidation both of Zn–Pb and Ag_2Zn_3 cannot be satisfactorily separated by liquation.

This tendency to oxidation has been counteracted by Edelmann and Rössler,[5] by the addition of 0.5 per cent. Al to the Zn, which resulted in a crust rich in Ag and poor in Pb. Their process has been abandoned,[6] as the presence of 0.1 per cent. Cu, of 0.05 per cent. As, and of 1 per cent. Sb neutralized the beneficent effect of Al; the presence of 0.03 per cent. Sb was harm-

[1] *Monatsheft für Chemie*, 1911, XXXII, 563, 597; *Min. Ind.*, 1911, XX, 493.

[2] *Zt. anorg. Chem.*, 1906, XLVIII, 347.

[3] *Internat. Zt. Metallogr.*, 1913, III, 145; *J. Inst. Met.*, 1913, IX, 214.

[4] *J. Russ. Phys. Chem. Soc.*, 1914, XLVI, 175; *J. Soc. Chem. Ind.*, 1915, XXXIII, 1211; *Chem. Abstr.*, 1915, IX, 777.

[5] *Berg. Hüttenm. Z.*, 1890, XLIX, 245, 249; 1891, L, 123; *Eng. Min. J.*, 1890, L, 573; 1891, LI, 404, 582; 1893, LVI, 245, 447, 568.
Schnabel-Louis, "Handbook of Metallurgy," 1905, I, 688.

[6] Schnabel, *Berg. Hüttenm. Z.*, 1897, LVI, 39, 40.

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less. Hasse[1] replaced the 0.5 per cent. Al by 0.3 per cent. Mg, and obtained results similar to those of Edelmann and Rössler.

There exists as yet no definite information as regards the solubility of Pb in Ag_2Zn_3 or of Ag_2Zn_3 in Pb.

2. *Gold.*—The constitutional diagram of the alloy series Zn-Au, drawn by Vogel[2] and given in Fig. 488, shows three chemical compounds, AuZn, Au_3Zn_5, and $AuZn_8$, with freezing-points at 744°, 651°, and 490° C., and no solid solutions. The compounds AuZn and Au_3Zn_5 furnish a satisfactory explanation for Au being taken up by Zn; in fact, it combines with Zn more readily than does Ag. It used to be common practice to add to the Pb-bath enough Zn to saturate the Pb and to remove all the Au with some Ag, and to treat this Au-crust separately from the Ag-crust. This method of working has been abandoned in the United States, but is still in operation in some European and Australian refineries.

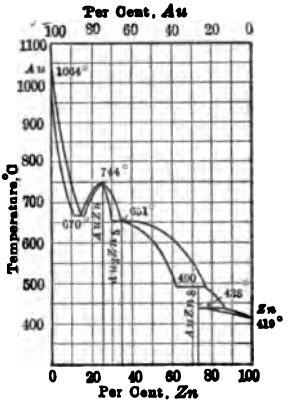

FIG. 488.—Alloy-series Zn–Au.

3. *Copper.*—The freezing-point curve of the alloy-series Zn–Cu of Shepherd[3] and Tafel,[4] reproduced in Fig. 489,[5] shows the existence of the compound Cu_2Zn_3, freezing at 830° C., which is probably the cause of Cu readily entering the crust. In fact Au and Cu combine with Zn before Ag. It may be noted that the alloys of Au and Cu[6] form solid solutions throughout. The Au-crust mentioned above is always richer in Cu than the subsequent Ag-crust.

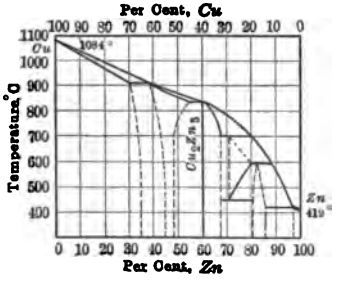

FIG. 489.—Alloy-series Zn–Cu.

Large-scale tests have shown that consumption of spelter increases with the Cu-content of the Pb; it becomes noticeable with 0.07 per cent. Cu. It is therefore of greatest importance to melt bars of lead bullion slowly in the reverberatory furnace or kettle, and to dross the lead carefully so that most of the Cu may be removed by skimming before the additions of spelter are made. In the endeavor to increase tonnage this important feature has often been overlooked. If liquid lead bullion is transferred from the blast furnace to a kettle, it is important to allow it to cool to the required degree. Some refiners remove from the surface all the dross that can be

[1] *Op. cit.*, 1895, LIV, 483; *Min. Ind.*, 1895, IV, 483; *Zt. Berg. Hütt. Sal. W. i. Pr.*, 1897, XLV, 323.

[2] *Zt. anorg. Chem.*, 1906, XLV, 323.

[3] *J. Phys. Chem.*, 1904, VIII, 421.

[4] *Metallurgie*, 1908, V, 349.

[5] Explanation of details, see Hofman, "Copper," 1918, p. 28.

[6] Roberts-Austen and Kirk-Rose, *Proc. Roy. Soc.*, 1901, LXVII, 105. Kurnakow-Schemtschuschny, *Zt. anorg. Chem.*, 1907, LIV, 164.

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skimmed readily, allow the lead to solidify, remelt slowly, and skim again. The solubility of Cu in Pb has been taken up in §8.

4. *Platinum and Palladium.*—Both metals are collected in the crust. According to Deville and Debray,[1] Pt forms the compound Pt_2Zn_3; they failed to find a similar compound for Pd.[2]

The entrance of Pt metals into the crust may also be favored by the solid solutions of Pt with Au,[3] Ag,[4] and Cu;[5] of Pd with Au,[6] Cu,[7] and Ag;[8] and of Pd with Pt.[9]

5. *Tellurium.*—According to Heberlein[10] this metal readily enters the crust. He found that with lead bullion containing 0.0025 per cent. Te, an addition of 0.25 per cent. spelter caused 95 per cent. of the Te to enter the crust. The curve of Kabayashi[11] shows the chemical compound TeZn, freezing at 1238° C., with eutectic points very close to the ends of the series.

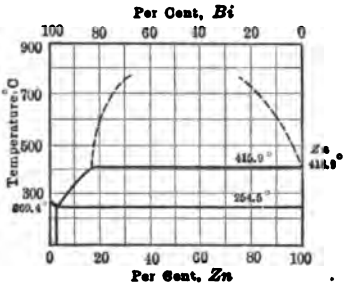

FIG. 490.—Alloy-series Zn-Bi.

6. *Bismuth.*—This metal does not enter the crust; the desilverized lead runs higher in Bi than did the original lead bullion. In one case[12] the Bi-content of the refined lead rose from the usual figure of 0.0008 to 0.0233 per cent. during a period when lead bullion running high in Bi was being desilverized. Another example is given in Table 119. The curve of Heycock and Neville,[13] supplemented by the researches of Spring and Romanoff,[14] and Arnemann,[15] given in Fig. 490, shows conjugate solutions, an eutectic with 2.8 per cent. Zn, freezing at 269.4° C., and a slight solubility of Bi in Zn. This explains the tendency of Bi to remain with the Pb and not to be taken up by the Zn.

7. *Arsenic.*—The curve of Friedrich and Leroux,[16] given in Fig. 491, shows that the two metals have no affinity whatever; hence As will not enter the crust. It

[1] *Ann. Min.*, 1859, XVI, 33, 51.
[2] See also *Compt. rend.*, 1882, XCIV, 1557.
[3] Doerinckel, *Zt. anorg. Chem.*, 1907, LIV, 347.
[4] Doerinckel, *loc. cit.*, 341.
Thomson and Miller, *J. Am. Chem. Soc.*, 1906, XXVIII, 1115.
[5] Doerinckel, *Zt. anorg. Chem.*, 1907, LIV, 337.
[6] Ruer, *op. cit.*, 1906, LI, 393.
[7] Ruer, *loc. cit.*, 225.
[8] Ruer, *loc. cit.*, 316.
[9] Puschin-Laschitschenko, *op. cit.*, 1909, LXII, 34.
Rudolphi, *op. cit.*, 1910, LXVII, 65.
[10] *Berg. Hüttenm. Z.*, 1895, LIV, 42.
[11] Kyoto, *Mem. Coll. Science, Imp. Univ. Kyoto*, 1911, III, 217; *J. Insl. Met.*, 1911, VI, 336.
[12] Junge, *Freiberg. Jahrb.*, 1895, p. 3.
[13] *J. Chem. Soc.*, 1897, LXXI, 394.
[14] *Zt. anorg. Chem.*, 1893, XIII, 29.
[15] *Metallurgie*, 1910, VII, 201.
[16] *Metallurgie*, 1906, III, 477.

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does, however, interfere with the entrance of other metals, and with the separation of the crust from the underlying lead. As little as 0.10 per cent. As in the lead shows its harmful effect. In skimming a kettle containing lead with 0.10 per cent. As, the crust is mushy and does not work well in the Howard press (§248). The skimmed lead does not have the usual beautiful, smooth, dark blue surface, but shows instead a rough, grayish white surface. If skimming is continued until the lead solidifies, the surface will hardly change in appearance. Experiments in a 40-ton kettle have shown that with lead bullion assaying 170 oz. Ag per ton and over, the presence of 0.10 per cent. greatly increased the amount of spelter required for desilverization. Arsenic appears to follow the Zn more readily than does Sb (see below). In one instance[1] the relative proportions of As and Sb in the lead bullion were 1 : 2.5, whereas in the retort bullion resulting from the distillation of the zinc crust, it was 1 : 0.25. Arsenic has to be removed from the lead bullion before any zinc addition is made; and luckily this is readily accomplished by the oxidizing fusion in softening (§241).

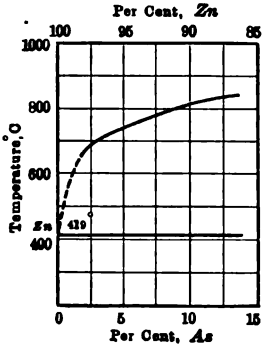

FIG. 491.—Alloy-series Zn–As.

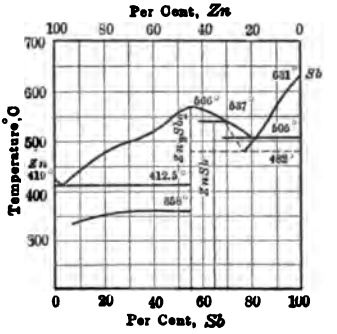

FIG. 492.—Alloy-series Zn–Sb.

8. *Antimony.*—The metal enters the crust in a small degree; most of it remains with the lead bullion. This is explained by the curve of Schemtschuschny,[2] given in Fig. 492, which shows the stable compound Zn_3Sb_2, freezing at 566° C., the unstable ZnSb, freezing at 537°, and two eutectics, one with 2.5 per cent. Sb, freezing at 412.5° C., the other with about 81 per cent. Sb, freezing at 505° C. The eutectics counteract the tendency of the compound Zn_3Sb_2 from entering the crust to any degree.

Some European works, which refine a uniform lead bullion, treat it directly as long as the percentage of Sb does not exceed 0.6 to 0.7 per cent. In the Harz Mountains lead bullion with 0.7 per cent. Sb and 41 oz. Ag per ton is desilverized directly with a zinc consumption of 1.3 per cent., and a yield of 81.34 per cent. market lead. The changes such bullion undergoes by drossing and by two zinc additions are shown in Table 119.[3]

[1] Heberlein, *Berg. Hüttenm. Z.*, 1895, LIV, 42.

[2] *Zt. anorg. Chem.*, 1906, XLIX, 386.

[3] Waldeck, F., "Streifzüge durch die Blei und Silberhütten des Oberharzes," Knapp, Halle, 1907, 57.

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TABLE 119.—CHANGES IN LEAD BULLION BY DROSSING AND ZINCKING

Metal, per cent.	Composition of lead bullion		
	After drossing	After first zincking	After second zincking
Sb	0.6627	0.6453	0.6328
Cu	0.0965	0.0402	0.0138
Bi	0.0009	0.0011	0.0013
Zn	0.0020	0.1873	0.7010
Ag	0.1400	0.1370	0.0004
Fe	0.0020	0.0020	0.0017

In the United States, refineries receive lead bullion greatly varying in its Sb-content. It is therefore the general practice to remove the Sb by an oxidizing fusion in softening (§241) before zincking. Antimony does not interfere with the work of the Howard press (§248) as does arsenic.

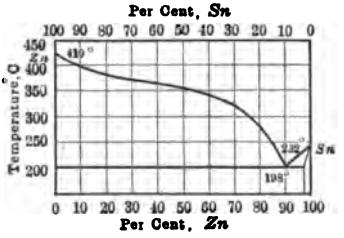

FIG. 493.—Alloy-series Zn–Sn.

9. *Tin.*—This metal remains with the lead, as can be seen from the freezing-point curve of Heycock-Neville[1] and Arnemann,[2] given in Fig. 493, which shows an eutectic with 90 per cent. Sn, freezing at 198° C., a solid solution of Zn in Sn to the extent of about 2 per cent., but no chemical compound. Tin interferes with zinc desilverization and has to be removed by softening the lead bullion (§241).

10. *Nickel-cobalt.*—These two metals occur very slightly in lead bullion; the small amounts present are completely removed by careful drossing so that they need not be considered in regular work. If drossing has not been well done, and some Ni and Co remain with the lead, they might enter the crust, as Ni forms the compound $NiZn_8$, freezing at 876° C.,[3] and Co the compound $CoZn_4$, freezing at 873° C.[4]

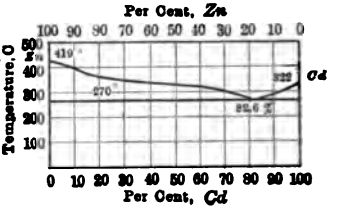

FIG. 494.—Alloy-series Zn–Cd.

11. *Summary.*—From the well-established behavior in zincking of the metals considered, and from the freezing-point curves discussed, the conclusion appears justifiable that only a metal which forms a chemical compound with Zn is likely to enter the crust, and that an eutectic acts in the opposite direction.

12. *Spelter.*—The spelter used for desilverization ought to be pure.

[1] *J. Chem. Soc.*, 1897, LXXI, 383.
[2] *Metallurgie*, 1910, VII, 201.
[3] Tafel, *Metallurgie*, 1907, IV, 784.
Voss, *Zt. anorg. Chem.*, 1908, LVII, 68.
[4] Lewkonja, *op. cit.*, 1908, LIX, 321.

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The effect of *Cadmium* is negligible. It forms, as seen in Fig. 494,[1] an eutectic with 82.6 per cent. Cd, freezing at 270° C.

The case is different with *Iron* which, as shown in Fig. 495,[2] forms two chemical compounds, $FeZn_7$ at 662° C., and $FeZn_3$ at 770° C., as well as a solid solution of Zn with 0.7 per cent. Fe. The presence of Fe in Zn is therefore harmful. Practical experience with Fe-bearing spelter by Jernegan,[3] Föhr,[4] the writer, and others has shown that the zinc consumption is much greater, if the Zn contains Fe. Thus Föhr found that he required four times the normal quantity of spelter with a brand containing Pb 2.75, Fe 0.61, Cu 0.077 per cent., Sn, As, Sb, Sd, S, and C traces. In analyses of good brands Fe appears only in the second decimal.

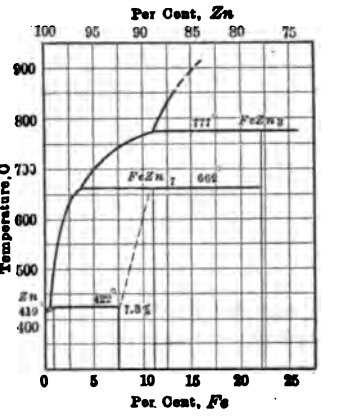

FIG. 495.—Alloy-series Zn-Fe.

13. *Conclusion.*—From what has been said regarding the behavior of the metals found in lead bullion and spelter, it is seen that both must be reasonably free from harmful metals, if desilverization is to proceed in a satisfactory way. Tests made by Kirchhoff[5] on lead bullion containing 4.5 per cent. foreign metals, such as Cu, As, Sb, Bi, and Zn, showed that 2.87 per cent. zinc was required to desilverize the lead when the bullion had not been softened, whereas 1.75 per cent. was sufficient, if softening had preceded the desilverization; the relative quantities of market lead produced were 43 per cent. and 72 per cent. of the lead bullion charged. Table 120 shows how the silver contents decreased with each addition of zinc. With the crude lead, the first five zinc additions served only to remove the impurities to such a degree that the desilverization could begin. That the first addition of zinc to the softened lead took up so little silver shows that the lead must have been very coppery.

The four metals that interfere principally with desilverization are Cu, Sn, As, and Sb; the first is sufficiently removed by drossing, the other three by an oxidizing fusion.

As regards spelter, most of the metal obtained from the smelting of ores runs sufficiently low in Fe to make it suited for the Parkes process; thus Prime Western with Pb 1.50 per cent. and Fe 0.06 per cent. is the grade ordinarily used. The zinc recovered from the treatment of drosses, scrap, etc., is usually worthless. An analysis for Fe answers the question of suitability in almost all cases.

[1] Heycock-Neville, *J. Chem. Soc.*, 1897, LXXI, 383.
Hindrichs, *Zt. anorg. Chem.*, 1907, LV, 417.
[2] Vegesack, *op. cit.*, 1907, LII, 37.
Arnemann, *Metallurgie*, 1910, VII, 201.
[3] *Tr. A. I. M. E.*, 1873–74, II, 288.
[4] *Berg. Hüttenm. Z.*, 1888, XLVII, 28.
[5] *Metallurgical Review*, 1877–78, I, 224.

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TABLE 120.—DESILVERIZATION OF NON-SOFTENED IMPURE LEAD BULLION

Number of additions of zinc	Lead bullion			
	Non-softened		Softened	
	Ounces silver per ton	Pounds of zinc	Ounces silver per ton	Pounds of zinc
After drossing	85.60	.....	94.90	.....
1	85.50	250	85.60	150
2	85.30	250	47.60	150
3	83.80	150	16.10	150
4	83.50	100	1.70	150
5	83.00	100	0.18	100
6	48.20	100	.....	.....
7	8.20	100	.....	.....
8	0.80	70	.....	.....
9	0.15	30	.....	.....

233. **Outline of Process.**—In the Parkes process the lead bullion, usually received in bars, is first melted and drossed, so as to remove all the harmful metals which can be separated by liquation, and then subjected at an elevated temperature to oxidizing influences to oxidize the remaining impurities and volatilize as well as scorify them with the litharge formed, whereupon they can be removed by skimming. The whole operation goes by the name of *Softening*. The common American practice is to soften the lead bullion in a reverberatory furnace; in some cases melting is carried on in a kettle, and liquated liquid lead bullion then transferred to a reverberatory furnace for the oxidizing fusion. In some European works a liquating reverberatory furnace is used for melting. Oxidation of liquated lead bullion at an elevated temperature in a kettle by stirring with steam is unsatisfactory, as the antimony skimming quickly corrodes the kettle, and the amount of skimming formed is excessive.

The softened bullion is discharged into a kettle in which it is desilverized.

The desilverized lead has to be freed from the 0.6 to 0.7 per cent. Zn which it retains, before it can be used in the metal industries. This operation, called *Refining*, is carried on either in reverberatory furnaces by means of an oxidizing fusion at an elevated temperature, or in a kettle by means of steam which at 900 to 1000° C. is decomposed by the zinc contained in the lead. The result of both operations is refined lead and an intermediary oxide product. The lead is molded into bars from the furnace direct or goes first into a market kettle. The intermediary products, *i.e.*, drosses, softening skimmings, and refining skimmings or oxides, are worked up separately.

The zinc crust, which contains considerable amounts of lead beside the zinc and precious metals, is distilled in retorts to recover as much zinc as possible, to be used again, and an enriched retort bullion, which is cupelled for silver (perhaps doré) and litharge. The intermediary products from distillation

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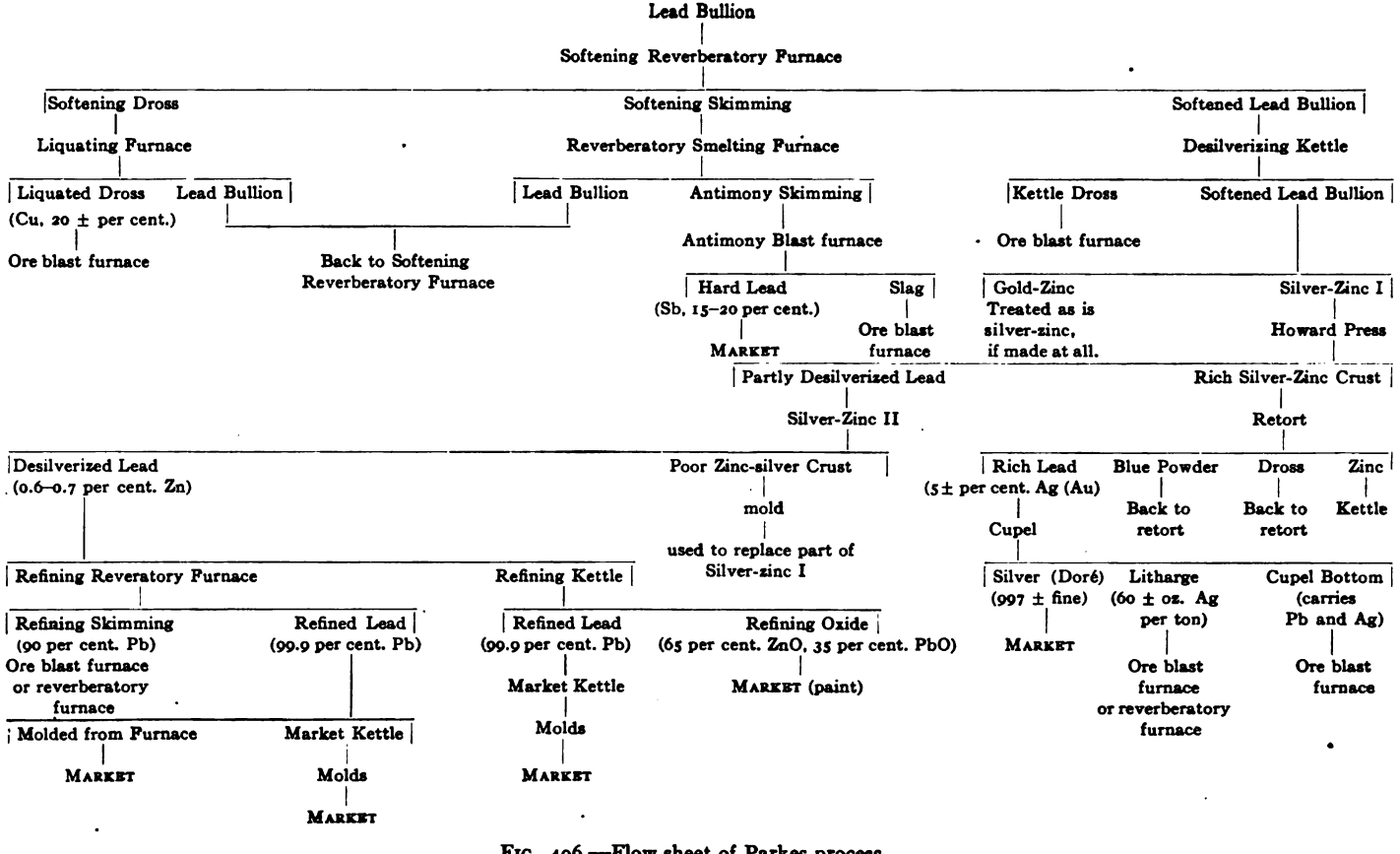

FIG. 496.—Flow-sheet of Parkes process.

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(blue powder and drosses) and cupellation (litharge and cupel-bottom) are worked up in a suitable manner.

This brief outline is shown with some detail in the flow-sheet given in Fig. 496.

234. Plant, General.—The general plan of a desilverizing plant varies somewhat with the location and the prevailing practice of operating. All arrangements, however, have to be of such a character as to require as little handling of lead and intermediary products as possible. In fact, the greater part of the lead, when once charged into the softening furnace, will not be handled again in a modern plant until it is ready for shipment. The result is that a vertical section through a refinery will show a number of terraces on which are placed the apparatus in the order required by the process. This is the case with three of the examples given below. Since the handling of molten lead by means of the Miller centrifugal pump has come into use, the number of terraces has been reduced, as shown in the fourth example.

Most European plants are altered Pattinson batteries and are therefore built on a horizontal plane. This arrangement[1] has advantages as regards primary cost and perhaps supervision and control work, but the operation cannot but be more expensive. Drawings of the refinery at Monteponi, Sardinia, have been published by Ferraris.[2] Below are given drawings of four terraced plants, which may serve to represent modern practice.

235. National Smelting and Refining Co. South Chicago, Ill.[3]—This plant was erected in 1889 by F. B. F. Rhodes; it was taken over in 1899 by the American Smelting and Refining Co., and underwent several changes before it was closed in 1914.[4] In Figs. 497 and 498 are given a vertical section and plan of the original plant. Though it is small, when compared with other works, having only two 30-ton kettles, and has liquating kettles which have become obsolete since the invention of the Howard press, its leading features were excellent when built, and are worth following today. On the highest level are two softening furnaces, which receive each 40 tons lead bullion and prepare it for the two 30-ton kettles. The latter are a stage lower, and there the softened bullion is desilverized. The apparatus for liquating zinc crusts is also upon the same floor. In the drawing, each desilverizing-kettle has close to it only one liquating kettle, with a small kettle for liquated lead, showing that no distinction is made between gold and silver crusts. If the Howard press be used, the liquating kettle or liquating reverberatory furnace will not be required. The liquated crusts pass from this floor into an adjoining building, placed to the right or the left of the main building, the floor of which is on a level with the scale-floor. It contains two departments—the retort-

[1] Roesing, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1888, XXXVI, 103; *Berg. Hüttenm. Z.*, 1888, XLVII, 337.

Saeger, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1893, XLI, 268; drwg. of Tarnowitz plant.

[2] *Oesterr. Zt. Berg. Hüttenw.*, 1905, LXXX, 455; *Eng. Min. J.*, 1905, LIII, 455.

[3] Pulsifer, *Min. Eng. World*, 1913, XXXIX, 153, 205.

[4] Pulsifer, *op. cit.*, 1915, XLV, 16.

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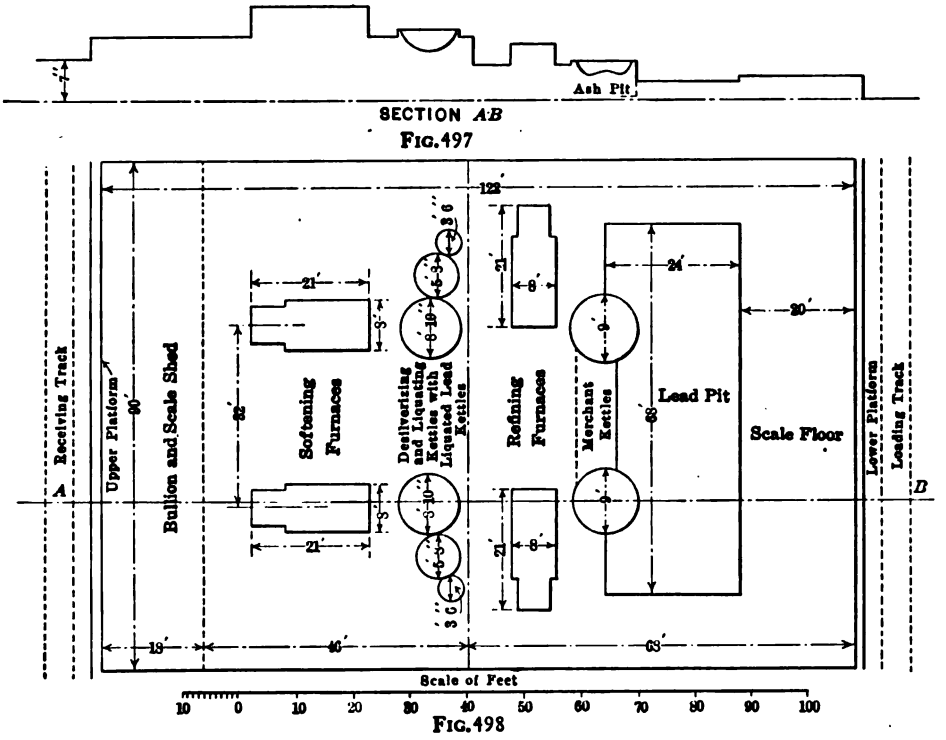
room, where the crusts are distilled, and the cupelling-room, where the retort-bullion is turned into silver, or doré-silver, bars. In the plant shown in the figure only doré bars will result. Following the desilverized lead, the refining furnaces are reached on the next level, in which the desilverized lead is de-zinckified. Thence it passes into the merchant-kettles, and from these into the molds placed in the lead-pit. The market lead is loaded on trucks on the scale-floor that are of the same construction as the bullion-receiving trucks; they are run on scales, and the weighed lead is transferred into the cars on the



FIGS. 497 and 498.—Section and plan, Parkes plant, National Smelting & Refining Co., South Chicago, Ill.

loading-track. The plant for working the by-products is placed on the side of the main building, opposite that where the zinc-crusts are treated. The manner of dealing with these varies greatly in different refineries, and will be discussed later on.

236. Selby Smelting and Lead Co., Selby, Cal.[1]—This plant was erected in 1884 and is in successful operation at present. A sectional elevation is given in Fig. 499. On the top floor are two 60-ton oil-fired softening furnaces, and two tracks on which travel the trucks which bring the lead bullion from the smelting department. The softening-furnace floor and the columns supporting

[1] Rickard, *Min. Sc. Press*, 1916, CXII, 505.

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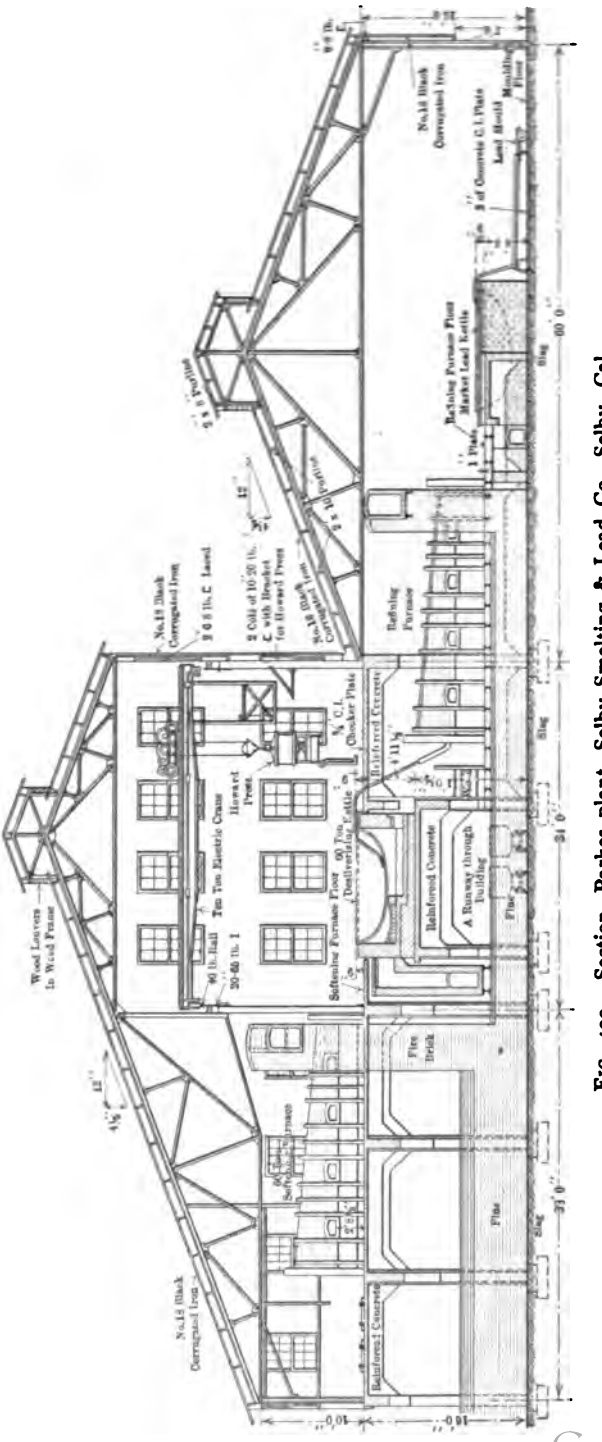
No.18 Black Corrugated Iron
Wood Louvres to Wood Frame
No.18 Black Corrugated Iron
No.18 Black Corrugated Iron
Ten Ton Electric Crane
Softening Furnace Floor
Softening Furnace
Reinforced Concrete
Reinforced Concrete
Reinforced Concrete
Refining Furnace
Refining Furnace Floor
Refining Furnace Market Lead Kettle
Flue
Flue
Flue
Flue Brick
Flue
Bldg
Bldg
Bldg

it are of concrete reinforced by corrugated bars. The softened lead bullion is tapped into troughs which deliver into four oil-fired cast-iron desilverizing kettles, 10 ft. in diameter and 3 ft. deep, each holding from 60 to 65 tons lead. The floor and supports are of reinforced concrete; the floor is covered with ¾-in. cast-iron checker plates. The four kettles are served by a 10-ton electric crane for the manipulation of the Howard press and the exchange of defective kettles. The desilverized lead is transferred by a 2½-in. siphon into two 60-ton oil-fired reverberatory refining furnaces. The refining furnace floor and supports are of reinforced concrete; the floor is covered with ¾-in. cast-iron checker plates; the supports are designed to carry a load of 200 lb. per sq. in. The part of the floor which is situated over the flue is made of 1-in. cast-iron checker

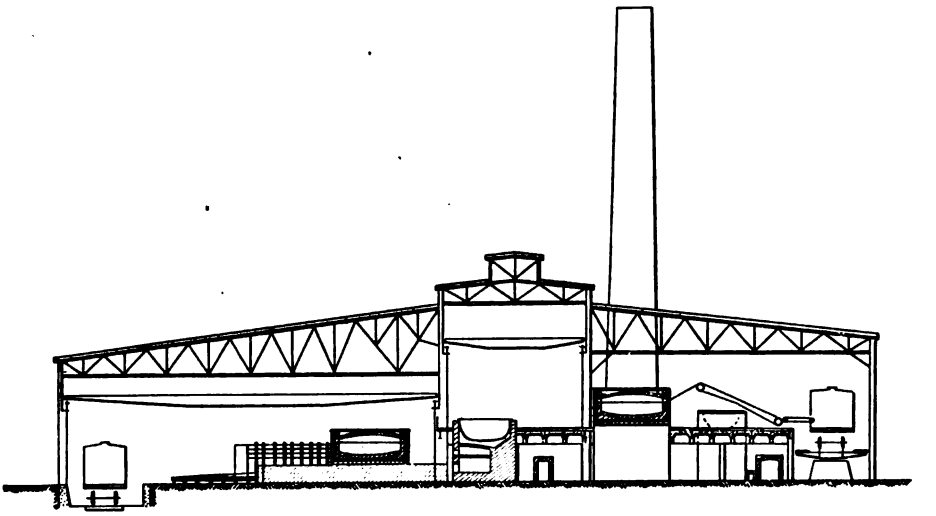

FIG. 500.—Cross-section main battery Parkes plant, International Lead Refining Co., East Chicago, Ind.

plates resting on I-beams as shown; the remainder is supported by 13-in. concrete walls. The refined lead is tapped into the single market-lead kettle of 60 tons capacity. Here the lead is allowed to cool and then siphoned into 50 molds placed in a three-quarters circle on the molding floor. A mold furnishes a 100-lb. pig. The molding or ground floor of the building has a slag foundation leveled with 3 in. of concrete and covered with cast-iron plates. The retorting and cupelling divisions are in separate buildings not shown in the section.

237. International Lead Refining Co., East Chicago, Ind.—This plant,[1] erected in 1912, has a monthly capacity of 5000 tons lead bullion. A cross-section through the main battery is given in Fig. 500, and a flow-sheet showing the auxiliary apparatus in Fig. 501. The flow-sheet is about the same as the tree in Fig. 496.

The site of the plant covers an area of 64 acres. The cars bringing the lead

[1] Hulst, *Tr. A. I. M. E.*, 1914, XLIX, 532.

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bullion arrive on a track provided with a 100-ton track scale; from the main track branch off five storage tracks furnishing space for 100 cars. The principal operations are carried on in the single main building, a steel structure, 480 by 180 ft. and 50 ft. high, with three levels, shown in Fig. 500, on which are broad-gauge

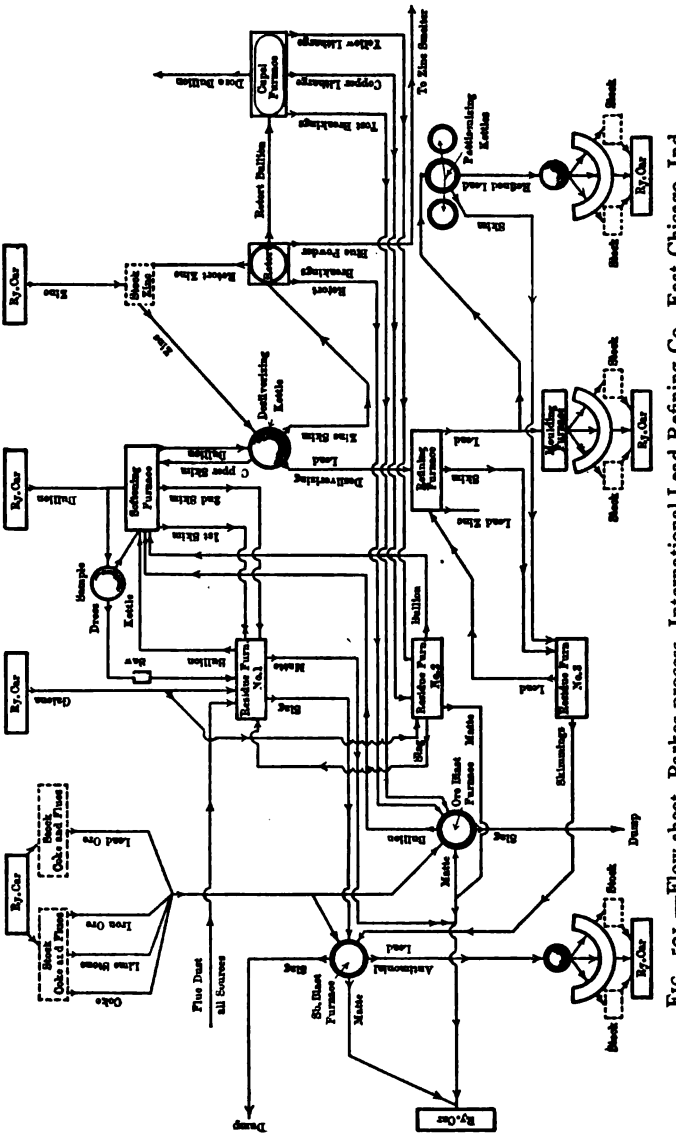

FIG. 501.—Flow-sheet, Parkes process, International Lead Refining Co., East Chicago, Ind.

tracks. Two 15-ton electric cranes of 28- and 77-ft. span, covering the kettle, and the refining and molding divisions, travel the entire length of the building.

The main battery contains two sample-kettles of 45 tons capacity; two 60-ton liquating kettles; two 300-ton softening furnaces (13 ft. 6 in. by 28 ft. 2 in.

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and 31 in. deep, Figs. 517–524); two 100-ton desilverizing kettles; one 300-ton refining furnace of the same size as the softening furnaces; and one 200-ton molding reverberatory furnace. For the treatment of intermediary products there are provided: three residue reverberatory furnaces (8 by 16 ft. and 20 in. deep, inside, Figs. 631–634) of 30 tons capacity for copper dross and antimony skimming, litharge, and silver-free refining skimming; two circular water-jacket blast furnaces (42 in. in diameter at tuyères, 14 ft. high, five 3-in. tuyères) of 40 tons capacity each for silver-bearing and silver-free antimonial intermediaries; eight oil-fired distilling furnaces with retorts holding 1300-lb. charges, Figs. 607–612; and two Rhodes cupelling furnaces with tests holding 5 tons rich lead, Figs. 657–664. The lead bullion from residue reverberatories Nos. 1 and 2, Fig. 501, is tapped into pots, taken by a crane to the two liquating kettles, and stored. On Saturdays the frozen lead bullion is liquefied, purified by compressed air, drossed, pumped into the softening furnace, and treated over Sunday, as, being hard, it requires more time than the regular 24 hr. given to softening.

In addition there is provided a small Pattinson division with a daily capacity of 150 tons lead for converting part of the refined lead, containing 0.01 to 0.12 per cent. Bi, by a single crystallization and the Hulst press (page 493) into corroding lead with not over 0.05 per cent. Bi. It contains two crystallizing kettles of 60 tons capacity, four heating kettles of 20 tons capacity, and one Hulst press.

The gases from the cupelling, residue, and blast furnaces are drawn through a brick and steel flue by an 8-ft. fan (American Blower Co.), driven by a 35-h.p. motor, and delivered into a bag house of brick and steel, 50 by 60 ft. and 50 ft. high. The bag house has four bays, each with 144 woolen bags, 18 in. in diameter and 30 ft. long.

Steam for the softening and refining furnaces, and for the air-compressor, is furnished by two Hawkes boilers; electric power is purchased; water is delivered from the city main to a 50,000-gal. tank elevated 50 ft.; waste water is collected in a 100,000-gal. sump and returned to the elevated tank.

There is a change-house, 35 by 85 ft., with toilets, wash-basins, lockers, and lunch room; and an office building and laboratory, 36 by 128 ft.

The lead bullion, which comes mainly from the company's smeltery at Tooele, Utah, contains Pb 99.5 per cent., Ag 65 and Au 0.4 oz. per ton. It is not sampled, but delivered direct from the cars into the softening furnaces by means of a steam-driven Howe conveyor, Figs. 504 and 505. Other lead bullion is unloaded into the two 45-ton kettles, melted, drossed, sampled, and then pumped into the softening furnaces. The dross is cast into molds, Figs. 353–356, and sampled by means of a circular saw (§176).

238. Balbach Smelting and Refining Co., Newark, N. J.—This plant, shown in horizontal and vertical longitudinal sections in Figs. 502 and 503, is the latest, having been erected in 1915. It differs radically from the other refineries discussed in that it has two levels instead of the usual three, and that it melts its lead bullion in kettles instead of in reverberatory furnaces.

The lead bullion is melted and drossed in the premelting kettles, and trans-

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ferred by means of a centrifugal pump into a trough delivering into the softening furnaces on the next level. The softened bullion is pumped back by the same means to the desilverizing kettles, and treated with zinc. The desilverized lead is transferred by pumping into one of the three refining furnaces, to be freed from the zinc it retains. The refined lead goes to the single receiving reverberatory furnace, from which it is molded direct. The lead from the upper refining furnace flows into the receiving furnace, that from the lower refiners is pumped into the receiver.

The horizontal section, Fig. 502, shows at the right the receiving tracks for lead bullion and materials, in the center the desilverizing division, and at the

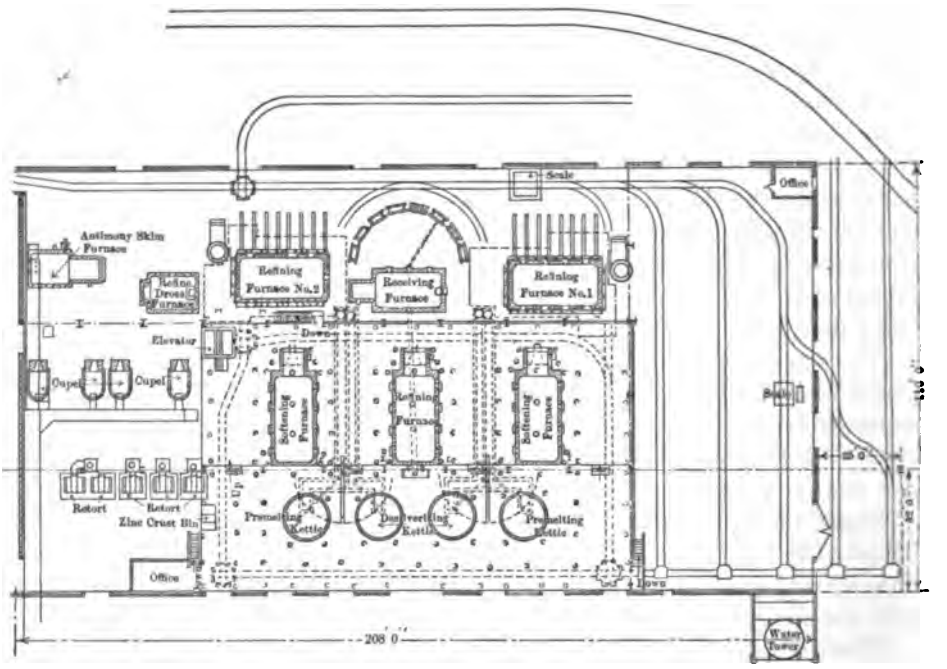

FIG. 502.—Horizontal section Parkes plant, Balbach Smelting and Refining Co., Newark, N. J.

left the auxiliary apparatus, reverberatories for dross and skimming (the blast furnace is in the smelting department), the retorts for zinc-crusts, and cupels for enriched lead. Farther to the left is the parting plant, not shown.

The lead bullion, arriving on the receiving tracks, is unloaded on to trucks, sampled, and raised with a traveling electric crane of 29-ft. 7-in. span by truck-loads of 50 bars, and delivered into the premelting kettles. The pigs of lead bullion on a truck are tied by a chain, and this connected with the hook of the crane.

The two pairs of 60-ton, coal-fired kettles on the upper platform intended for premelting and desilverizing are 10 ft. 3 in. in diameter and 3 ft. 4 in. deep. Under normal conditions, *i.e.*, when the brick walls and kettles are hot, it takes

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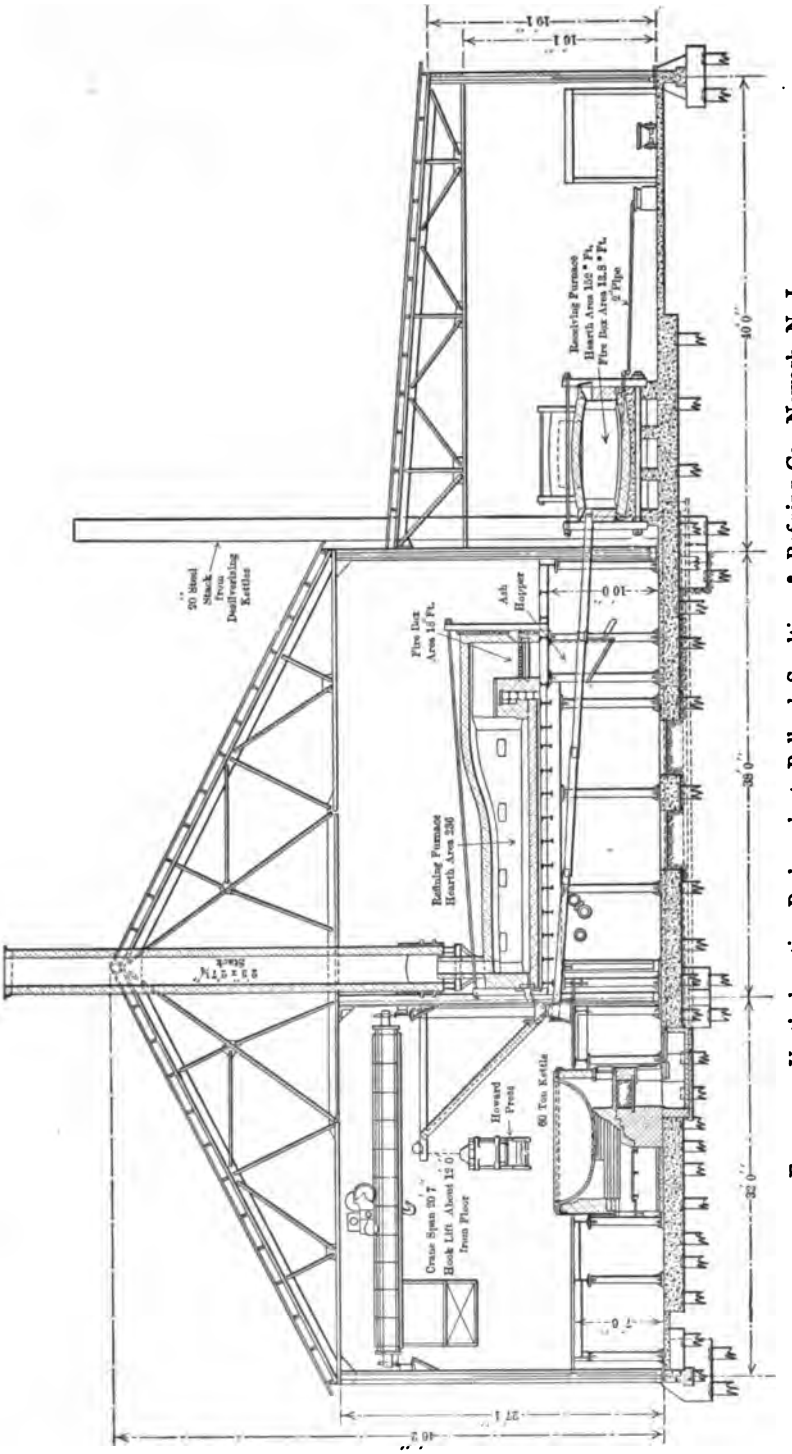

FIG. 503.—Vertical section Parkes plant, Balbach Smelting & Refining Co., Newark, N. J.

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6 hr. to melt a charge of 60 tons lead with a coal consumption of 1 ton. A melting kettle lasts for 228 charges; a desilverizing kettle has to be replaced after 150 charges have been treated.

When the lead bullion has been melted, the dross is skimmed and freed from excess lead by two Howard presses working with a pressure of 100 lb. per sq. in. These are suspended from jib cranes, each serving two kettles. The squeezed dross is dropped on to the floor, the discharges being separated from one another by pieces of scrap sheet iron in order to facilitate their removal.

The drossed bullion from two kettles is pumped in about 15 min. into one of the softening furnaces, which holds 120 tons lead, and has a hearth 10 ft. 10 in. by 22 ft. and 1 ft. 4 in. deep. The softened bullion is pumped into the two 60-ton desilverizing kettles. These are worked together by three men, who desilverize their charges in 12 hr.

The desilverized lead is now pumped into one of the three refining furnaces which are of the same size as the two softening furnaces.

The receiving furnace is smaller in area than the other reverberatories; it has a hearth 9 by 17 ft. 1 in. and 3 ft. 7½ in. deep, and holds 100 tons lead.

The retort room is well ventilated; it has at present eight oil-fired retorts (1200-lb. charges) in a row, each with its own chimney; two retorts have a single hood to carry off the fumes.

Six cupels, with tests 6 ft. 4 in. by 4 ft. 5 in. and 6 in. deep, holding 6500 lb. lead, have a solid foundation. The working bottom consists of a course of magnesite or chrome brick, 4½ in. thick; on top of this is rammed the usual limestone-clay mixture reaching to the working door; above this the walls are of fire-brick.

The Cottrell plant, which receives the gases from six cupelling and silver furnaces, is intended to treat 45,000 cu. ft. gas per min. It has three compartments, each with 30 steel pipes, 10 in. in diameter and 16 ft. long. The current is 0.15 amp. at 75,000 volts. Its work is very satisfactory; no fumes whatever are seen to arise from the stack.

239. Operation, General.—In the flow-sheet, Fig. 496, giving an outline of the Parkes process, there are brought out the leading steps necessary for desilverization. These are, receiving lead bullion, softening of lead bullion, desilverizing softened bullion, refining desilverized lead, molding refined lead, treatment of zinc crusts, and treatment of intermediary products. Each step requires certain operations, all of which demand a detailed discussion.

240. Receiving Lead Bullion.—The lead bullion usually arrives at the refinery in car loads of from 25 to 50 tons. In most instances the bullion is weighed and sampled at the refinery; in some works these operations are carried out at the smeltery, especially if both establishments belong to the same company. The necessity or omission of sampling may govern the unloading of a car.

If the bullion is to be weighed and sampled at the refinery, the receiving track, Fig. 497, is laid so low that the bottom of the car is on a level with the "upper platform" of the works. Along the whole length of this platform, and

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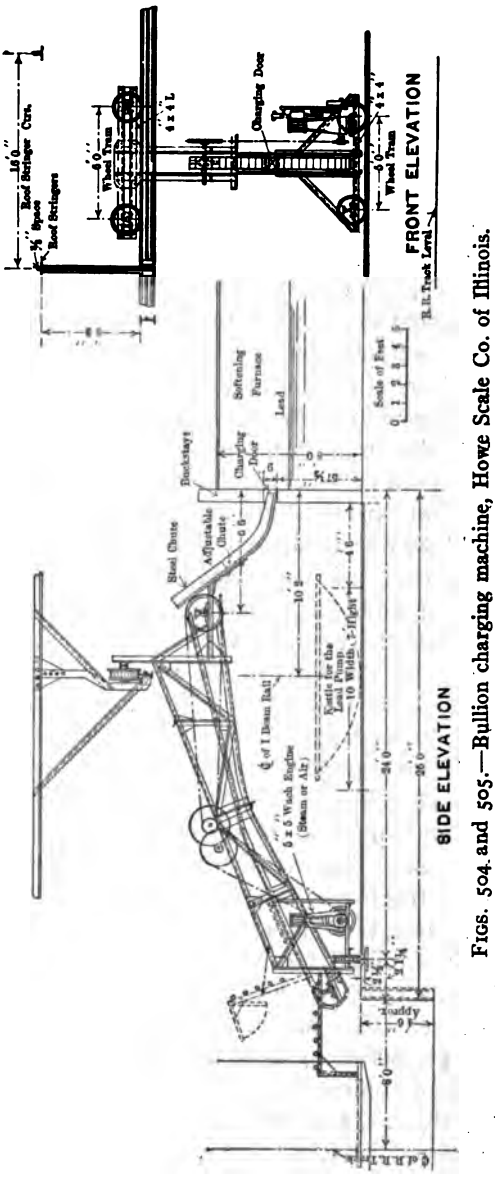
parallel with the railroad, runs a narrow-gauge track of from 16 to 22-in. gauge, which bears a number of strongly built low bullion-trucks. They may be built as follows. A frame, consisting of two pieces of channel-bar iron 3 ft. long, is fastened to the two axles of the wheels, and steadied by two iron bands running diagonally. The bullion is carried out from the car and loaded upon the truck standing before the car-door; when this is filled to a height convenient for lifting (about 3½ ft.), it is moved on, and another takes its place. The trucks are run on scales placed at one or two points in the "bullion and scale-shed," the bullion is weighed, and is then sampled with punches from one truck directly upon another, which then moves straight to the softening-furnaces, or to places near them, and no more handling is required before the bullion is charged into the furnaces. The bullion produced in the smelting department of the refinery is loaded at the blast furnaces or reverberatory furnaces, on to the same kind of truck, brought by an elevator to the "bullion and scale-shed," and then passes to the softening furnaces.



FIGS. 504 and 505.—Bullion charging machine, Howe Scale Co. of Illinois.

At some works the bullion is unloaded and weighed as described, and then transferred to a 45- or 60-ton kettle, melted in about 6 hr. when the kettle is hot, and drossed; dip samples are taken from the drossed bullion, and saw-samples from the dross which has been cast into bars (Fig. 365). The molten bullion is then siphoned (Fig. 500) or pumped (Fig. 503) into the softening furnace. A similar procedure is found at some works which transfer their blast or reverberatory furnace bullion in pots to the receiving kettle. When this is filled, and the lead been held at the correct

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temperature of 370 to 380° C. for sufficient time to permit the dross to rise, the latter is removed, freed from adhering lead by a Howard press, and the bullion transferred to the softener.

Plants which do not sample the bullion, usually check the weights furnished by the smeltery and the railroad by weighing car and bullion on a track-scale, and then unload the car into the softening furnace by means of the conveyor manufactured by the Howe Scale Co. of Illinois, Chicago, Ill. This charging machine, indicated in Fig. 500, was first used at the National plant in South Chicago.[1] It is shown in side and front elevations in Figs. 504 and 505. The leading parts are a steel structure, carried at the lower end by a wheel tram running on a track close to the railroad car, and suspended from a similar tram at the upper. The structure forms the support of a steel conveyor driven by a 5 by 5 Wach engine. The car bringing the bullion arrives on a depressed track, is spotted so as to bring its door approximately in line with the charging door of the softening furnace; the charging machine is moved so as to be in line with the doors of the car and the softener, its tilting roller platform is lowered, and the engine started. Four unloaders in the car pick the bars of bullion from their piles in the car, deposit them on the roller table, and push them onto the roughened tables of the conveyor, which carries them up the incline, and delivers them into an adjustable chute down which they glide through the charging door, and land toward the center of the softening furnace. In order to prevent piling up of bars, it is important that the furnace be hot at the start and contain a bath of lead which shall melt the bars as quickly as they are charged. The new cold charge will in time cool the bath of lead and the furnace to a sufficient degree for the dross to rise and carry with it most of the Cu contained in the bullion. In from 2½ to 3 hr., 2500 and even more bars of lead bullion are thus readily transferred from the railroad car into the softening furnace.

In some plants, where there is not sufficient room for a charging machine, hand-charging has been replaced in part by mechanical devices. The lead bullion is piled in the car on a rectangular frame of heavy plank placed near the door. The loaded frames are removed one after another by an electric truck, transferred to a place beneath an overhead electric crane which, by means of a chain, picks up the bullion from the frame and delivers it on to one or two inclined roller tables placed before the charging doors of the softening furnace. From a table two men deposit on to the rollers the pigs, which glide into the furnace. They handle 40 tons in an hour.

241. Softening Lead Bullion.—The object of softening is to separate from the lead bullion produced in the blast furnace, impurities, such as a Cu, S, Sn, As, Sb, etc., that would interfere with the desilverization. It comprises two processes, liquation and oxidation. By the former, metals and their compounds held in solution by the red-hot blast furnace lead are separated again from the readily fusible lead by melting it slowly at a low temperature. By the latter, metals alloyed with the lead, and more easily oxidized than the lead, are removed by heating it to a bright-red with access to air, with the result that these metals

[1] Pulsifer, *Min. Eng. World*, 1913, XXXIX, 154.

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are converted into oxides, which are volatilized to some extent, but mostly combine with the PbO formed at the same time, and are drawn off either as a powder or as a slag from the surface of the metallic lead.

When lead bullion is melted slowly in a softening furnace at the low heat of 370 to 380° C., there rises to the surface a dark colored, half-melted, pasty substance, the furnace-dross, consisting of a mixture of Pb, Cu, S, As, etc. The slower the melting, the more effectual will be the separation of the Cu from the Pb. The analyses given in Table 121 show the purification effected in the lead bullion by liquating, and the compositions of the drosses when they have been freed as much as possible from adhering lead.

TABLE 121.—ANALYSES OF LEAD BULLION BEFORE AND AFTER DROSSING, AND OF DROSS

	Clausthal, (*a*) Harz Mount.		Lautenthal, (*a*) Harz Mount.		Freiberg, (*b*) Saxony		Denver, (*c*) Colo.		Port Pirie, N. S. W. (*d*)	Montemey, Mexico (*e*)	Hoboken-les-Anvers, Belg. (*f*)
	Before drossing	After drossing	Before drossing	After drossing	Before drossing	Liquated dross (5 per cent.)	Dross, before liquating	Dross, after liquating	Dross	Dross	Dross
Pb....	98.92944	99.0239	98.96475	99.1883	96.667	62.40	53.0	50.0	77.5	91.9181	67–70
Cu....	0.1862	0.1096	0.2838	0.0907	0.940	17.97	18.2	26.8	8.0	2.8500	14–16
Cd....	Trace	None	Trace	None	.....	.....	.....	.....	.....	.....	.....
Bi....	0.0048	0.0050	0.0082	0.0083	0.066	None	.....	Au, 0.30 oz.	.....	N. d.	.....
Ag....	0.1412	0.1420	0.1413	0.1440	0.544	0.17	.....	75.0 oz.	.....	0.8122	0.2–0.4
As....	0.0064	0.0053	0.0074	0.0032	0.449	2.32	.....	7.31	0.7	2.2340	5–8
Sb....	0.7203	0.7066	0.5743	0.5554	0.820	0.98	.....	0.18	0.7	1.3400	2–4
Sn....	None	None	None	None	0.210	0.04	.....	.....	.....	Au 0.0038	0.004
Fe....	0.0064	0.0042	0.0089	0.0048	0.027	0.43	.....	.....	1.0	0.0521	0.5–2.0
Zn....	0.0028	0.0017	0.0024	0.0015	0.022	0.07	.....	.....	1.2	.....	.....
Ni....	0.0023	0.0017	0.0068	0.0038	0.0055	1.09	.....	.....	.....	Trace	.....
Co....	0.00016	Trace	0.00035	Trace	.....	.....	.....	.....	.....	Trace	.....
S.....	.....	.....	.....	.....	0.200	4.00	2.0	3.6	.....	0.7598	.....
O.....	.....	.....	.....	.....	.....	1.87	.....	.....	.....	.....	.....
Slag, ash, hearth-material, Insol.					.....	8.66	1.8	4.8	2.1	.....	5–7

(*a*) Hampe, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1870, XVIII, 203. (*b*) Schertel, *Berg. Hüttenm. Z.*, 1882, XLVI, 293. (*c*) Iles, private notes, August, 1896. (*d*) Delprat, *Tr. Australas. Inst. Min. Eng.*, 1907, XII, 21. (*e*) Hahn, *Tr. Inst. Min. Met.*, 1899–1900, VIII, 273. (*f*) Gov't. Report, *Ann. Min. Belg.*, 1901, VI, 255.

Table 122, published by Blakemore,[1] gives the assays of lead bullion and ensuing dross. Similar data have been given in Table 84.

In liquating,[2] the metals which form solid solutions with lead will remain with the lead; those which are insoluble or form chemical compounds will separate and remain with the dross; those which form eutectic mixtures will be liquefied at the liquation temperature before the lead; as the temperature is well above the melting-point of lead and therefore of the eutectic, they will be found to have been collected in the lead.

[1] *Tr. Australas. Inst. Min. Eng.*, 1898, V, 225.

[2] Friedrich, *Metallurgie*, 1906, III, 13.

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TABLE 122.—ASSAYS OF LEAD BULLION AND ENSUING DROSS

Lead bullion			Dross		
Ag, oz. per ton	Au, oz. per ton	Pb, per cent.	Ag, oz. per ton	Au, oz. per ton	Cu, per cent.
143.03	12.25	63.10	67.36	6.00	5.1
168.68	12.00	66.06	53.46	5.66	3.8
117.49	12.92	60.70	41.31	1.32	1.8
118.04	12.48	69.10	39.86	1.32	1.5
121.16	14.78	73.50	60.66	1.49	3.6
105.30	11.28	54.10	38.24	0.54	1.9
117.42	13.52	72.70	53.67	3.53	3.9
114.08	11.95	76.60	52.40	4.92	6.4
81.12	6.61	67.60	43.77	6.76	1.8
115.30	11.26	60.70	37.86	6.78	3.3
148.54	16.94	68.80	58.14	5.86	1.5
220.02	21.98	73.50	62.24	6.48	1.4
179.40	42.28	82.40	60.96	10.00	1.5
139.32	16.08	62.40	95.56	16.00	6.1
340.04	37.38	62.20	85.34	8.40	1.9
157.34	16.52	70.80	27.98	2.86	1.4
141.53	14.30	71.00	31.20	3.36	1.1
121.74	12.94	78.40	53.08	6.00	1.6
132.72	12.90	76.10	60.06	7.50	1.7
246.38	24.04	60.60	114.14	20.44	9.3
154.52	13.34	75.10	48.56	5.44	2.6

In Table 121, the analyses from Clausthal and Lautenthal demonstrate that the character of a comparatively pure lead is improved by melting slowly in a kettle and then drossing. By comparing the two Freiberg analyses, the degree to which the foreign matter of a very impure lead bullion may be removed will be seen; viz., nearly all the S, 96 per cent. Cu, 99 per cent. Ni and Co, 25 per cent. As, and only 5.8 per cent. Sb, and 1.54 per cent. Ag; Bi remained entirely in the liquated lead, and all the Sn excepting 0.9 per cent. These data find their explanations in the freezing-point curves. It must be remembered, however, that dross always retains entangled particles of lead bullion.

On melting the dross from Freiberg in a crucible, Schertel[1] obtained in well-separated layers, the three products lead, speiss, and matte, given in Table 123.

TABLE 123.—ANALYSES OF PRODUCTS FROM MELTING DROSS

	Lead	Speiss	Matte
Ag	0.34	.....	.....
Cu	1.79	37.60	47.70
Pb	96.50	25.68	32.80
Ni	0.08	8.60	0.25
As	0.75	27.00	1.15
S	.....	.....	17.72

[1] *Berg. Hüttenm. Z.*, 1882, XLI, 293.

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The absence of Fe in either speiss or matte proves that these impurities do not result from finely divided blast furnace speiss or matte dissolved in the lead, as has been often thought. It tends to show that, being held in solution or suspension, the impurities unite, on liquating, to form compounds that are not fusible at the temperature at which the lead was melted, and that the concentration of Cu in dross is due probably to the presence of S and As and not to the separation of an alloy of Pb and Cu.

Table 122 shows that the Au-content of the dross rises with that of the Cu. This finds its explanation in the fact that Au and Cu form solid solutions throughout.[1]

In melting lead bullion that is very rich in Cu, the amount of dross formed is sure to be large. As it carries with it precious-metal-bearing Pb, it will reduce the direct yield from the charge below a practical limit. In order to prevent this, at some works galena is added to the softening-furnace charge for the purpose of forming a matte with the Cu contained in the lead bullion. The result is that all the Au remains in the metal bath and only a small percentage of Ag is carried off by the matte.

Heberlein[2] found that Te readily entered the dross and did not follow the As and Sb into the skimming as one might be led to expect from the similarity between it and these metals. This may be attributed to the chemical compound PbTe which freezes at 917° C.[3] or 904°,[4] and may be assisted by the presence of Cu which forms the compounds Cu_4Te_3 and Cu_2Te, freezing at 620 and 856° C.[5]

At Pertulosa, Italy, in refining lead bullion with 0.0025 per cent. Te, the lead products contained the amounts of Te shown in Table 124.

TABLE 124.—TELLURIUM CONTENTS OF REFINERY PRODUCTS

Products	Cu, per cent.	As, per cent.	Sb, per cent.	Te, per cent.	Ag, per cent.	Au, per cent.
Softening dross	34.29	.....	.....	0.0525	.....	.....
Softening skimming	0.76	1.36	8.42	0.0185	.....	.....
Softened lead bullion	0.23	0.051	0.12	0.0159	0.522	0.0012
Softened bullion (after first zincking)	.....	.....	.....	0.0019	.....	.....
Softened bullion (after second zincking)	.....	.....	.....	0.0008	.....	.....
Market lead (after third zincking and refining)	.....	.....	.....	0.00065	.....	.....
Refining skimming	.....	.....	.....	0.0010	.....	.....

[1] Roberts-Austen and Kirke-Rose, *Proc. Roy. Soc.*, 1901, LXVII, 105.
Kurnakow-Schemtschuschny, *Zt. anorg. Chem.*, 1907, LIV, 164.

[2] *Berg. Hüttenm. Z.*, 1895, LIV, 41; *Min. Ind.*, 1895, IV, 480.

[3] Fay-Gillson, *Tr. A. I. M. E.*, 1901, XXXI, 527.

[4] Kimura, *Mem. Coll. Science, Imp. Univ. Kyoto*, 1915, I, 419; *J. Soc. Chem. Ind.*, 1915, XXXIV, 1211.

[5] Chikashige, *Zt. anorg. Chem.*, 1907, LIV, 54.

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If a sample, taken from the lead after drossing, be poured into a small mold, and allowed to cool slowly, a crystalline, bright, pewter-white spot will appear on the slightly depressed, dull, grayish-white surface, which, in addition to the hardness of the lead, is characteristic for the presence of As and Sb.

If, after drossing, the temperature of the lead bath is raised to a good red heat and air freely admitted, the three leading impurities contained in the lead will oxidize in the following order: first Sn, then As, and lastly Sb; at the same time some Pb will be converted into PbO. The surface at first will become quickly covered with dark yellow skimming, which varies from powdery to pasty, but is not fused, on account of the refractory nature of SnO and SnO_2. It forms the so-called tin skimming, which consists mainly of antimoniate and stannate of lead and antimoniate of tin, and is worked by itself (§263).

As soon as the tin skimming has been drawn from the surface of the lead, this begins to give off fumes of arsenic and antimony, and arseniate and antimoniate of lead begin to form. A sample of this fume gave Iles: As 2.56, Sb 32.73, Pb 20.60 per cent., Ag 7.0 and Au 0.05 oz. per ton. The arseniate of lead is lemon-yellow to light brown, the antimoniate dark brown to black; both are fused and drawn off together as antimony skimming after the furnace has been sufficiently cooled to cause it to solidify. Toward the end of the operation the antimony oxide in the skimming will be replaced by lead oxide until the black color has changed to the greenish yellow of litharge.

Samples are then taken to see how far the softening has progressed. Before the Sb has been removed, a sample of the bullion taken in a ladle will "work," *i.e.*, small particles of melted black skimming will float on the surface of the lead, with a rotary motion which resembles that of particles of dark grease on hot water. As the softening approaches the finishing point, the globules become less in number and smaller in size, a thin coating of yellow litharge forms more readily on the red-hot lead, and finally no more globules are seen and litharge forms quickly.

When a sample of softened lead bullion is poured into a mold, allowed to cool slowly, and skimmed with a flat piece of wood, it will, when it has been solidified, have lost the characteristics of As and Sb, and the surface of the bar will have assumed a rich indigo-blue color. The lead has become so soft that it can be easily scratched with the finger nail; and incision made into a bar will show a bright metallic luster; cupelling a sample will furnish a cupel surface free from incrustations.

242. Softening Furnace.—The furnace used almost exclusively in the United States is the reverberatory; it serves for the operations of liquating and oxidizing fusions. Formerly softening furnaces were built large enough to hold from 8 to 10 per cent. more lead bullion than the kettle into which they discharged their contents. The kettles used to hold 30 tons softened and the reverberatories 40 tons crude bullion. As the time required for softening a large charge in a suitable furnace is not longer than a small one, but more economical in labor, fuel, and repairs, the tendency has been to increase the sizes of furnaces, and discharge their contents into several kettles. The desilverizing kettles have

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been increased to hold 60 and even 100 tons of lead, and the softeners to treat 300 tons. Softeners holding 90 tons could be charged by hand. With mechanical charging came the possibility of having larger furnaces. Large softeners were in operation in England before they were used in the United States; thus Hutchings[1] records a softener of 140 tons capacity.

Drawings of 30-, 60-, and 300-ton furnaces are given in Figs. 506–524. They represent the softener of the original National works, Figs. 506–512, for 40 tons; of the Selby Works, Figs. 513–516, for 60 tons; and of the International Works, Figs. 517–524, for 300 tons lead bullion. The drawings of the International Works contain also the details in which the refining reverberatories differ from the softening.

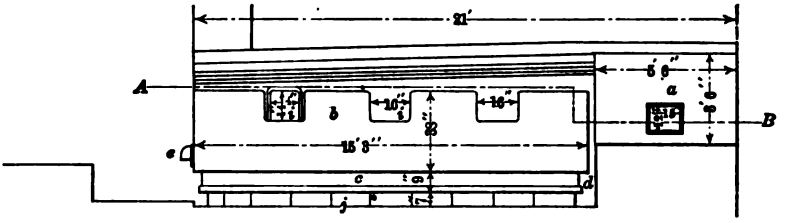

FIG. 506.—Elevation of skimming side.

FIGS. 506 to 512.—Coal-fired 40-ton softening furnace, National Smelting & Refining Co., South Chicago, Ill.

All furnaces have rectangular hearths, with length : width = 2 : 1, and are built of fire-brick enclosed in a boiler-iron pan; in section the hearth is dish-shaped. The greatest depth of the metal bath shows a range of from 12 to 16 in.; with the 300-ton furnace, it became necessary to increase the depth to as much as 31½ in. A deep hearth is good for drossing, one that is shallow is

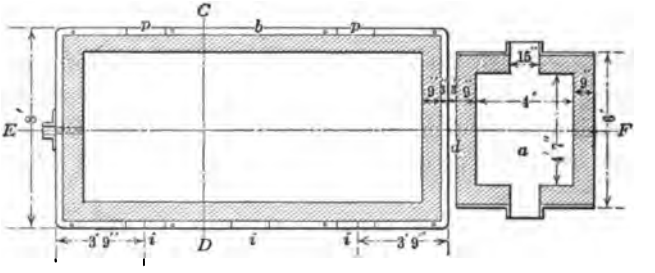

FIG. 507.—Horizontal section on line A-B.

favorable for oxidizing; as the latter operation takes up most of the time given to softening, hearths are made shallow. The hearth usually has a slope toward the tap-hole which ranges from 1 to 5 in. A furnace tapped at the side requires, of course, a smaller slope than one tapped at the end.

The pan holding the brickwork used to be of cast iron. In spite of all precautions taken to relieve it from strains, it usually cracked after it had been in use for a little while. At present it is built up from riveted boiler-iron plates.

[1] *Eng. Min. J.*, 1895, LIX, 26.

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The brickwork of the hearth is readily attacked by the antimony skimming and litharge formed in the oxidizing fusion. In order to counteract the corrosion, 2.5- and 3-in. water-pipe was placed next to the pan at the level of the lead-line. The brickwork was eaten away to a thickness of from 2 to 3 in.

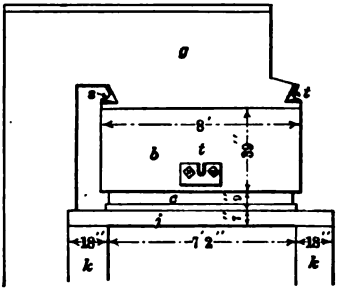


FIG. 508.—Elevation of flue end.

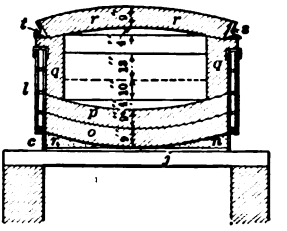


FIG. 509.—Vertical section on line C-D.

as quickly as without the pipe, but then the corrosion proceeded only very slowly, and the life was thus greatly prolonged. The use of water-pipe is not uncommon in cupelling furnaces. In 1873 E. Balbach Jr.,[1] enclosed the pan holding the hearth with another, leaving a water-space of 3 or 4 in. between

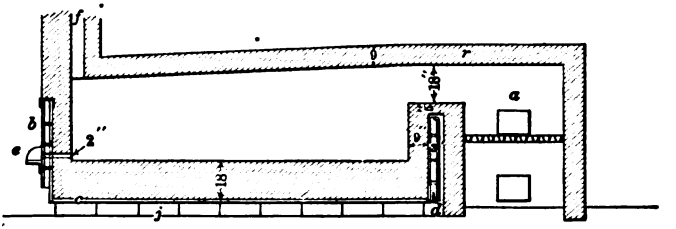
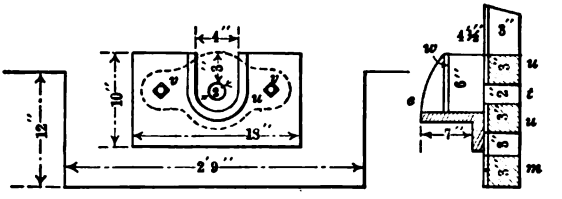


FIG. 510.—Vertical section on line E-F.

the two, and thus cooled not only the sides, but also the bottom. While this mode of cooling is very effective, there is too much of it, considering the fuel required to maintain the temperature necessary to soften the lead in a given time. It does check the corrosion of the hearth bottom by whatever litharge



FIGS. 511 and 512.—Detail of tap.

remains in the furnace after the lead bullion, or in the refining furnace the refined lead, has been tapped, and is not floated up again by the next charge. Air-cooling has always been found sufficient for the bottom; water-cooling is

[1] U. S. Patent, No. 141912, Aug. 19, 1873.

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necessary for the sides only; all softening and refining furnaces are provided with soft-steel jackets on sides and ends.

In order to increase the life of the brickwork, the fire-brick have been replaced, especially along the lead line, by magnesite and chrome brick.[1]

The manner of putting in the hearth varies somewhat. A layer of brasque is tamped in and then cut out so that the course of fire-brick, laid endwise, shall bring it to the desired shape and give it the necessary inclination toward the tap-hole (Fig. 521). In many furnaces only the central part of the inverted arch is thus carried by brasque and the remaining space is bricked up (Figs. 523 and 524). In putting down the bottom, the bricks have to be joined as tightly as possible. For this purpose they must first be carefully selected and fitted by rubbing together until all roughness is removed. Each brick is dipped into water and then into a clay mortar having the consistency of very thin gruel; it is then put in place and driven with a hammer against the brick it is to face. This makes the joint as close as possible and prevents the passage of lead. The sides of the furnace are built with the same care as the bottom. Commonly they rest on the curved working bottom to prevent this from rising. The roof is supported on either side by skewbacks, and the furnace bound with buckstays and tie-rods. The manner of tapping the furnace deserves special mention. The tap-hole is an opening, 2 to 3 in. in diameter and usually conical, in a cast-iron plate, which sometimes is water-cooled. The hole is closed by means of a clay plug rammed in firmly. In order to make breaking away impossible, the clay plug is reinforced by an iron plug held in place by an iron wedge driven between it and a horizontal piece of flat-iron kept in position by the vertical buckstays of the lead-spout.

The fuel commonly used for firing is bituminous coal (Figs. 506–512); in many works this has been replaced by oil (Figs. 513–516); in one plant provision is made for both coal- and oil-firing (Figs. 517–524). With a good grade of bituminous coal, natural draft is sufficient to soften the lead bullion in the required time. With slack coal, undergrate blast becomes necessary. The admission of air under pressure through a series of small pipes in the roof is satisfactory with coal running high in volatile matter. In using oil, the pressure-air serving as atomizer furnishes all the O necessary for oxidizing the impurities to be scorified.

243. Method of Working.—The mode of operating the softening furnace is regulated in such a manner that the time required for charging, softening, and tapping shall not exceed 24 hr. There is no difficulty in doing this with bullion containing about 1 per cent. impurity. The aim is to make up the charges in such a way that they shall not exceed this amount.

The bullion is charged by hand or mechanically; some or even all of it may be pumped in. In hand-charging, paddles are used to place the bullion. The paddle is a rectangular iron bar about 8 ft. long, made of 1⅛-in. iron, one end being flattened out for a distance of 2 ft. 6 in. to the width of 3 in., to receive a bar of bullion; the other is rounded off and bent to a ring.

[1] Havard, *Eng. Min. J.*, 1908, LXXXVI, 802.

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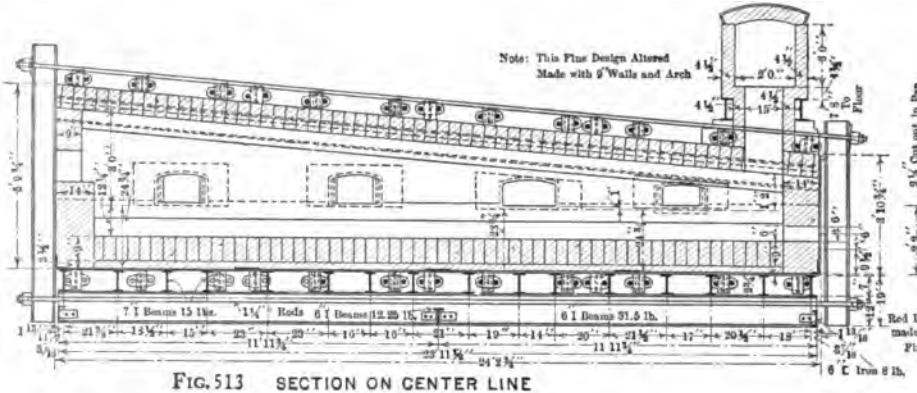
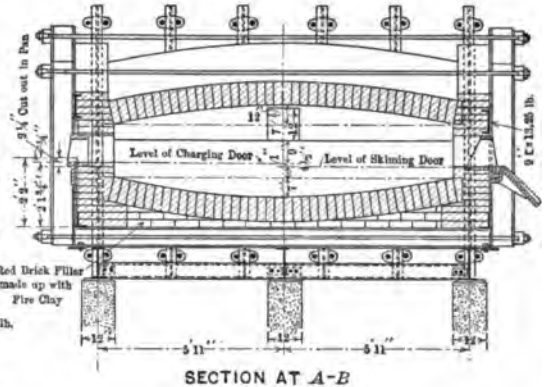


FIG. 513 SECTION ON CENTER LINE



SECTION AT *A-B*

FIG. 515

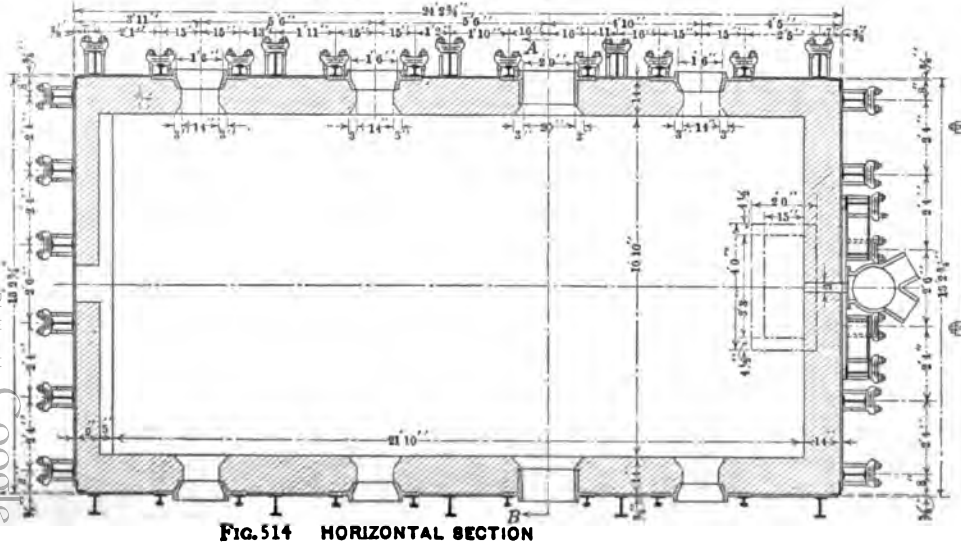
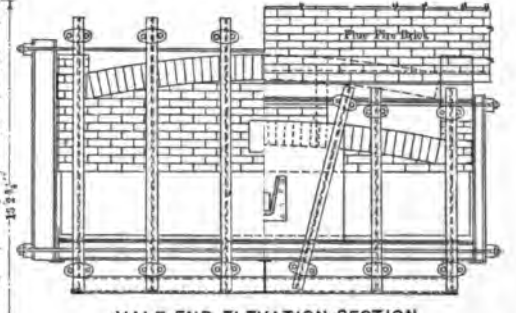


FIG. 514 HORIZONTAL SECTION



HALF-END ELEVATION SECTION

FIG. 516

FIGS. 513 to 516.—Oil-fired 60-ton softening furnace, Selby Smelting and Lead Works, Selby, Cal.

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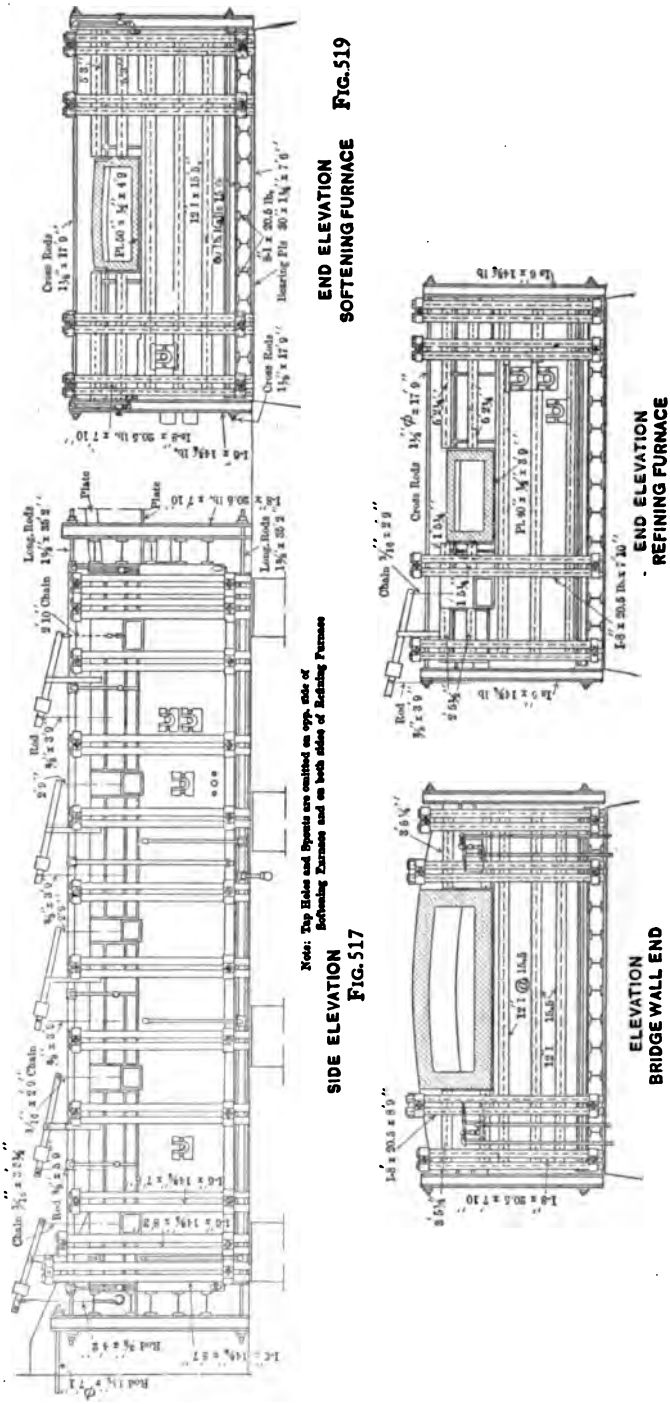

FIGS. 517 to 520.—Elevations.

FIGS. 517 to 520.—Coal- and oil-fired 300-ton softening and refining furnace, International Lead Refinery, East Chicago, Ind.

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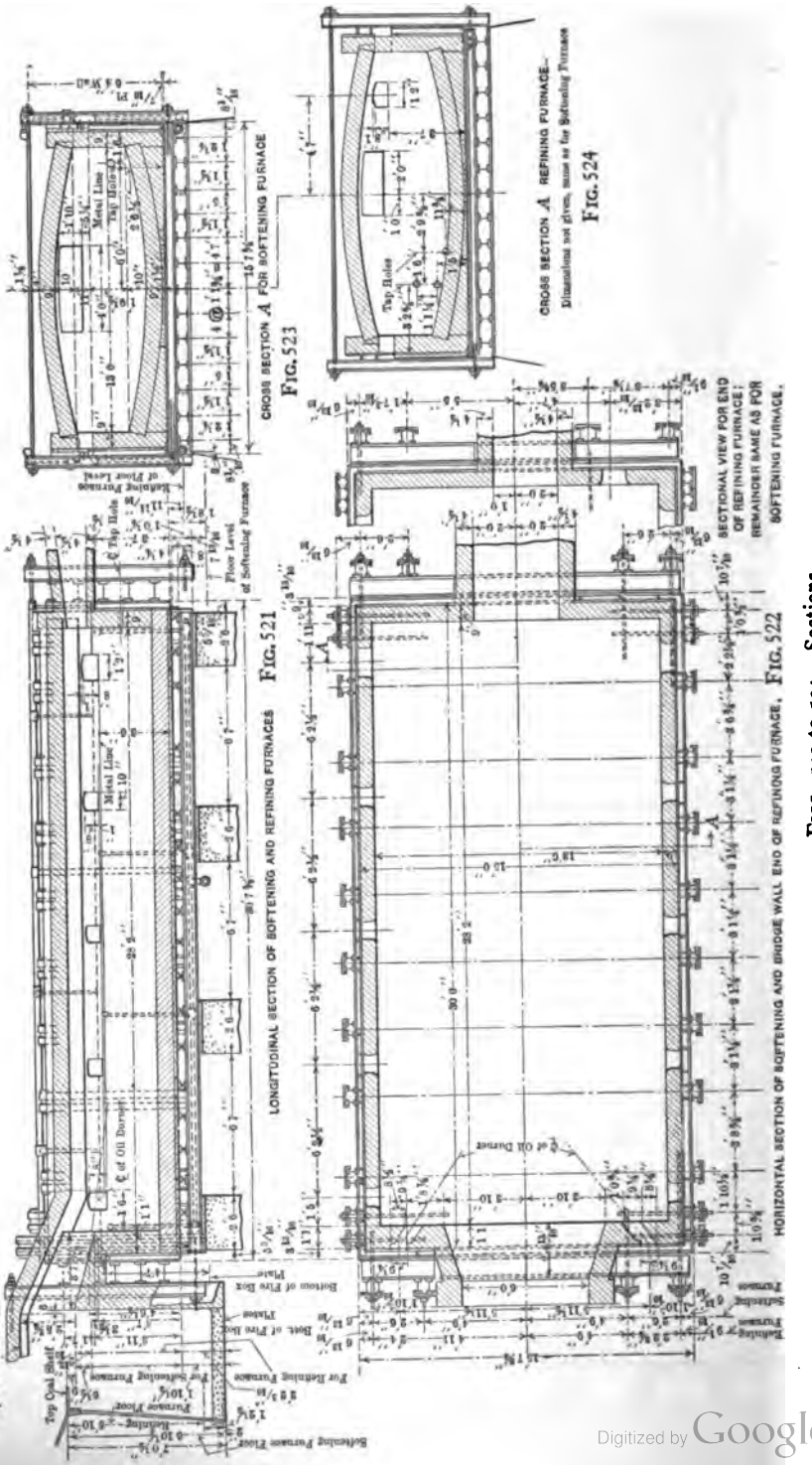
CROSS SECTION A FOR SOFTENING FURNACE
FIG. 523
CROSS SECTION A REFINING FURNACE
Dimensions not given, same as for Softening Furnace
FIG. 524
LONGITUDINAL SECTION OF SOFTENING AND REFINING FURNACES
FIG. 521
SECTIONAL VIEW FOR END OF REFINING FURNACE; REMAINDER SAME AS FOR SOFTENING FURNACE.
HORIZONTAL SECTION OF SOFTENING AND BRIDGE WALL END OF REFINING FURNACE.
FIG. 522

FIGS. 521 to 524.—Sections.

FIGS. 517 to 524.—Coal- and oil-fired 300-ton softening and refining furnace, International Lead Refinery, East Chicago, Ind.

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A lug is often cast on either side of the door-frame to support a roller. It serves as bearing for the paddle instead of the door-frame, and thus facilitates the manipulation. Charging by machinery has been discussed in §229.

The work of the Miller centrifugal pump is taken up in Figs. 563–566.

It is common practice in our days to dross very carefully the blast furnace bullion in a kettle before casting it into bars which are to go to a refinery. This explains the prevalence of hastening the melting of the bullion in the softener, which would be absolutely wrong with undrossed material. After the bullion has been melted, it is stirred to detach some of the lead held in suspension by the dross; sometimes fine coal is spread over it and stirred in. This is very effective when the bullion is pure, so that little dross rises to the surface. With impure bullion considerable fuel is required to have any effect, and there is danger of the temperature becoming too high, and the lead taking up again some of the impurities that had separated. The dross is removed by a rabble or a rectangular skimmer.

The head of the rabble is made of ⅜-in. iron, and is 3 by 12 in., the handle of ⅞-in. iron, and 10+ ft. long. The handle of the skimmer is of the same length and thickness as that of the paddle; the perforated part is made of ¼-in. iron, and is 10 by 12 in., the perforations being ⅜ in. in diameter.

With either tool about the same amount of lead is withdrawn with the dross from the furnace, so there is little choice between them; some prefer one, some the other. The handle often rests in a hook suspended by a chain from the roof, thus facilitating the work. The operator removes from one side the dross, which his helper on the other side collects with a rabble, pushing it toward the door or upon the skimmer. The dross, while being removed from the furnace, is collected in a slightly conical cast-iron mold running on wheels, *e.g.*, 2 by 3 ft. at the base and 14 in. deep, made of ⅝-in. iron, or in a slag-pot with tap-hole, or in an iron two-wheel barrow with perforated bottom, in order that some of the lead carried out with the dross may run off on the cast-iron plate in front of the skimming-doors and be returned to the furnace. The use of galena with bullion rich in copper has already been referred to above.

The dross drawn off is weighed, and a sample is taken from the lead remaining in the furnace to be assayed. The weight and assay-value of the bullion charged being known, the weight of the dross and the assay of the residual hard lead give the data necessary to calculate the total silver contained in the dross, and with it its assay. The amount of dross formed varies; it is about 3 per cent. of the bullion charged, and assays about 80 per cent. Pb. It is freed from some of its lead in a liquating furnace, and will then have a composition shown in Table 122.

The tin skimming that forms on raising the temperature after drossing is removed in the same way as the dross.

Most lead bullion contains so little arsenic that arsenic skimming is not kept separate from the similar antimony skimming. If there is enough arsenic present to warrant keeping the two skimmings separate, the arsenic skimming is taken and handled as is the antimony skimming.

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With the antimony skimming it is customary to cool the furnace by throwing open the doors, in order that the antimoniate of lead floating on the surface may harden, and be then taken off in the form of a thin crust by means of a hoe and a paddle. In withdrawing the skimming, it is advisable to leave in the furnace that part which adheres to the brickwork. The skimming eats into the brickwork for a short distance and then soaks more or less into the remainder. A crust of skimming adhering to the soaked bricks protects them from further attack.

If the bullion is very hard, skimming once will not be sufficient to soften it. The furnace is therefore again heated up; and as soon as the surface of the lead is well covered, cooling and skimming are repeated. Ordinarily two operations are sufficient, but sometimes three are necessary. To hasten the cooling of the furnace, slaked lime is sometimes spread over the metal bath. Some refiners add lime to the furnace after drossing, with the idea that an antimoniate of lime is formed, and thus less lead is oxidized during the softening. This effect of lime still remains to be proved. Any addition of lime to the furnace has the great disadvantage of interfering with the subsequent treatment of the antimony skimming, and is therefore better dispensed with altogether.

With bullion rich in Sb it is often not possible to finish the work of the furnace in 24 hr. An addition of refining skimming (§§251 and 252) or lead oxide from the molding apparatus greatly assists the oxidation of Sb. Ordinarily it is not advisable to use litharge from the cupelling furnace, as this is likely to contain Cu which would enter the bullion and affect desilverization in the kettle. Of course, litharge free from Cu is preferable to other lead oxides, as its Ag is taken up by the bullion. An addition of from 10 to 15 tons lead oxide to 250 tons drossed lead bullion is a common amount. The finer the oxide, the more quick will be its action.

Steam has been used to stir up the lead and thus expose fresh surfaces to the oxidizing action of the air. This is done by introducing through the charging doors a 1-in. pipe, to the end of which are screwed, by means of a T, two pipes having a number of perforations on either side and closed at the ends. The main pipe is bent so that it is introduced into the furnace and held in place by the closed furnace-door, which has been weighted; the two pipes at its ends will be pressed down into the lead, and run parallel to the sides of the furnace. Though the introduction of steam does shorten the time required for softening, it has the disadvantage that it forms a large amount of skimming, and that the swash of the lead oxide and antimoniate strongly corrodes the sides of the furnace. It has, therefore, not found much favor.

Another method to be mentioned is the one in use at Freiberg, where bullion rich in tin, arsenic, and antimony is softened. Blast is introduced on either side of the firebridge; the skimmings are removed at the flue end of the furnace as fast as they form. The tool used is a long iron hook, to which is fastened a triangular piece of wood, say 8 in. long. With it the skimmings drawn out of the furnace in a thin stream. In order to facilitate the work, and to enable the workmen to pass gently over the surface, and thus remove only

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skimmings, and no lead, the handle is supported by a hook suspended from the roof.

In this country blast is rarely used, as the walls of the furnace are corroded much more quickly than under normal conditions.

After the last skimmings have been removed, the doors are thrown open to cool the lead to a visible red before it is tapped into the kettle. The stream of lead is so directed as not to strike the bottom of the kettle. The skimmings are weighed and a sample of the softened bullion in the furnace is taken for assay; thus the silver contained in the skimmings can be calculated. The amount of skimmings formed is about 5 per cent. of the weight of the bullion charged.

The fuel consumed for the entire softening used to be about 156 lb. of soft coal per ton of bullion charged, or 8.67 gal. reduced oil, air being used as atomizer. The figure for coal equals 8.7 per cent. of the weight of the bullion charged or, assuming 80 per cent. market lead, 10.9 per cent. of the market lead. Hutchings[1] gives from 10 to 12 per cent. of the market lead; Huntington[2] gives 6.6 per cent. of the market lead with a 40-ton and 5 per cent. with a 43-ton furnace, using Newcastle coal, as averages at the works at Pertulosa, Italy. In this country, with a 300-ton furnace, the coal consumption has fallen to 2 per cent. (Hulst).

One man and helper attend to the work of the furnace.

A new furnace bottom absorbs a considerable amount of lead bullion; a 300-ton furnace takes up about 30 tons. One peculiarity still needs a satisfactory explanation, namely, that a larger proportion of Au than of Ag collects in it, considering the average composition of the bullion treated.

Schertel[3] found imbedded near the tap-hole of a softening furnace, the lining of which consisted of raw and burnt fire-clay rammed down firmly, a felt-like scaly metallic mass of a bright copper color which showed upon analysis: Cu 59.75, Pb 33.46, Sb 0.37, Ni 0.67, Ag 0.13, Au 0.0235, total metal 94.3925 per cent.; clay 5.6 per cent. Calculation upon a metal basis gives Cu 63.31, Pb 35.45, Sb 0.39, Ni 0.69, Ag 0.137, Au 0.025 per cent. The drossed lead bullion contained Cu 0.06 to 0.10 per cent.; this value is increased to 0.12 to 0.15 per cent. after the arsenic and antimony skimmings have been removed. This concentration of Cu is remarkable, as is that of Ni and Au, which are present in the lead bullion in exceedingly small amounts. Assays of antimony skimmings are given in Table 125.

The gases from the softening furnaces are conducted through flues and dust chambers to the stack. Experiences with a bag house have been unsatisfactory in that the meshes of the filtering cloth became clogged with hydrocarbons. Samples of flue-dust from softening furnaces show values given in Table 126.

[1] *Eng. Min. J.*, 1895, LIX, 171, 266.
[2] *Loc. cit.*, 291.
[3] *Freiberg. Jahrb.*, 1900, p. 59.

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TABLE 125.—ASSAYS OF ANTIMONY SKIMMINGS

	Hoboken-les-Anvers, Belgium(1)	Port Pirie, N. S. W.(2)	Port Pirie,* N. S. W.(3)		
Pb	50–65	74.0	51.80	48.20	48.70
Ag, oz. per ton	30–90	.....	12.91	15.24	18.27
Au, oz. per ton			1.41	1.70	1.94
Sb	15–28	10.7	.....	.....	.....
As	0.1–0.5	1.0	.....	.....	.....

* Includes 1 to 2 per cent. metallics.

(1) *Ann. Min. Belg.*, 1901, VI, 255. (2) *Tr. Australas. Inst. Min. Eng.*, 1907, XII, 21. (3) *Op. cit.*, 1898, V, 228.

TABLE 126.—ANALYSES OF FLUE-DUST FROM SOFTENING FURNACES

Pb	Zn	S	SO_3	As	Sb	Ag, oz.	Au, oz.	Authority
52.2	2.8	25.1	12.2	Trace	0.56	76.6	0.16	M. W. Iles.
33.8	4.4	.....	.....	.....	.....	30.0	0.16	M. W. Iles.
15.9	.....	.....	.....	.....	.....	9.1	0.16	F. B. F. Rhodes.

244. Desilverization of Softened Lead Bullion, General.—From the softening furnace the lead, when cooled to a visible red, or about 500° C., is tapped into the desilverizing kettle, which has been white-washed with lime water and heated to the point where a splinter of dry wood thrown on the bottom will ignite readily, which corresponds to about 400° C. The white-washing facilitates the removal of silver crusts which adhere to the sides when the kettle is cooling. If the lead were tapped into a cold kettle, this would be likely to crack on the bottom, and the time for bringing the lead up to the required temperature would be unnecessarily prolonged. The lead runs into a trough of cast iron, ⅜ in. thick, or a channel iron or an I-beam, placed beneath the discharge-spout of the furnace. In order to decrease the amount of dross, the lead runs from the trough through a cast-iron pipe which conducts it to the side of the kettle.

The kettle-dross formed amounts to about 1 per cent. of the bullion charged. It is skimmed off and added to the next charge in the softening furnace after the furnace-dross has been taken off. The kettle is now ready for the addition of zinc.

The quantity of zinc necessary varies with the purity of the lead and increases on the whole with the amount of Ag present. Presupposing the lead to be practically pure, it will take up (§19) and hold in solution at about 400° C. between 0.6 and 0.8, average 0.7, per cent. Zn. This amount has to be added to the kettle before the lead will give up any of its Ag; it is therefore a constant. The quantity of zinc required to combine with Ag must be based on that needed to form Ag_2Zn_3. Plattner[1] showed by experiments on a working scale, how the zinc required increased with the Ag-content. His data are given in Table 127.

[1] *Berg. Hüttenm. Z.*, 1889, XLVIII, 117.

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TABLE 127.—ZINC REQUIRED FOR DESILVERIZATION WITH INCREASING AG-CONTENT OF LEAD

Ag in lead bullion, oz. per ton	Per cent. of zinc added
28.09	1.34
111.56	1.84
148.16	1.96
245.00	2.45

Roswag[1] formulated the amounts of zinc required: $Z=23.22+0.223A$, in which Z = the zinc per 2000 lb. lead, and A = assay Ag oz. per ton. He also expressed algebraically the quantities of zinc found by Illing[2] to be necessary in desilverization: $Z=20.78+0.24A$, the letters having the same meanings as above. These general formulæ probably give an approximate idea of the total amounts required to desilverize lead bullion running low in Ag, say 30 to 50 oz. per ton; for rich bullion they are too high.

In practice it has been found to be impossible to desilverize high-grade lead bullion by a single addition of zinc; the richer the bullion the less difference is there between the assay values of the zinc-silver crust and the residual lead. Formerly it was customary to have three and even four zinckings. At present it is uniform practice in the United States to desilverize by means of two additions of zinc provided that no gold crust is made. This mode of operating may require more zinc than the one that prevailed formerly, but it is a decidedly cheaper method. The aim is to concentrate as much Ag (and all the Au) as possible into the first, the saturated, crust, and thereby utilize all the desilverizing power of the zinc. This crust, assaying over 2000 oz. Ag per ton, goes to the retorts. In the second, and final, zincking an excess of zinc over that which is required to combine with the Ag is used; this is necessary if the Ag is to be removed completely from the lead. The resulting unsaturated crust has still some desilverizing power; it is therefore collected in molds and used to take the place of part of the fresh zinc required in the first zincking. It is assumed that two-thirds of its zinc content is available as new zinc. The second crust assays from 10 to 30 oz. Ag per ton.

The analyses of Table 128 show the compositions of some zinc-silver crusts obtained from low-grade bullion, which has been drossed only and retains the Sb, and of liquated zinc-silver (gold) crusts, from softened bullion rich in Ag and Au. Of the zinc-silver-gold crusts, the one analyzed by Jenks (*h*) shows an abnormal composition, as the retort bullion assays ordinarily 2000 oz. Ag and 5 or 6 oz. Au per ton. This crust resulted from bullion containing as impurities Cu 0.37, As 0.02, and Sb 0.15 per cent. It looked like small sponges of brass floating in the normal gold crust. They formed to the amount of several hundred pounds.

All the German silver crusts given run much lower in silver than any from American refineries. Those from Altenau and Lautenthal retain more lead than any of the others, as they are to be melted again in a kettle and decomposed by steam, which could not be satisfactorily done if the liquation had been carried any further.

[1] "La désargentation de plomb," Dunod, Paris, 1884, p. 241.

[2] *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1868, XVI, 51.

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TABLE 128.—ANALYSIS OF ZINC-SILVER CRUSTS

	Altenau (a)	Lautenthal (b)	Mechernich (c)			Friedrichshütte (d)		Bleiberg (e)	Freiberg (f)	Denver (g)	Newark (h)	Selby (i)	Port Pirie (j)
	1880	1880	1875 (c)	1884 (d)	Average 1884 (d)	1869	1884	1901	.....	1896	1895	1916	1907
Pb	75.675	77.820	48.80	49.70	45–48	34.66	81.2	86.0	52.200	67.00	31.38	64	70
Zn	11.78	12.11	39.00	34.00	35–45	20.19	12.15–10.15	10.5	39.700	13.20	22.05	23	20
Cu	1.12	0.82	5.33	6.00	2–3	.....	.....	0.667	2.680	4.40	45.82	3	10
Ag	1.855	2.420	1.22	1.75	1.5–1.7	1.21	1.075–1.202	1.7–2.5	4.015	9.96	0.380	10	.....
Au	.....	.....	.....	.....	.....	.....	.....	.....	0.0153	0.0103	0.2774		.....
As	Trace	Trace	.....	.....	.....	.....	.....	—0.045	.....	.....	.....	.....	.....
Sb	.....	.....	0.36	.....	.....	0.83	.....	—0.400	.....	.....	.....	.....	.....
Cd	Trace	Trace	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Ni	Trace	Trace	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Fe	.....	.....	1.28	.....	.....	0.96	.....	Trace	.....	.....	.....	.....	.....
PbO	4.75	4.00	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
ZnO	0.60	0.44	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Bi_2O_3	1.72	0.37	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Sb_2O_3	0.63	0.98	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Fe_2O_3	1.87	1.04	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

(a) *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1880, XXVIII, 262. (b) *Ibid.* (c) *Berg. Hüttenm. Z.*, 1875, XXXIV, 129; *Eng. Min. J.*, 1877, XXIII, 161. (d) *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1886, XXXIV, 92. (e) *Ann. Min. Belg.*, 1901, VI, 251. (f) Schnabel-Louis, "Handbook of Metallurgy," 1905, I, 663. (g) Iles, private notes, 1896. (h) Jenks, private notes, 1895. (i) *Min. Sc. Press*, 1916, XCII, 505. (j) *Tr. Australas. Inst. Min. Eng.*, 1907, XII, 24.

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245. Gold-Zinc.—In the United States gold crusts are not made. The Au is taken out with the Ag, the doré silver crust is retorted, the retort bullion cupelled, and the doré silver parted either with sulphuric acid or electrolytically. The main reasons for this procedure are, that making a gold crust takes time, and requires the separate storing and handling of two classes of products; and the fact that the lead bullion usually, or perhaps frequently, retains small amounts of Au in spite of the greatest care given to the operation. It is not an uncommon experience to find no Au in using 1 A.T. for cupelling and parting; but with 4 to 6 A.T., Au puts in an appearance.[1] The disadvantages of making a gold crust, even when all the Au is satisfactorily recovered, make the procedure more expensive than taking out the Ag with the Au, and parting all the Ag instead of only part of it.

One important fact in making a gold crust is of general interest; it is that lead bullion will give up its Au and most of its Cu before the lead is saturated with zinc, whereas with Ag this is not the case. Table 129 shows this for a

TABLE 129.—ZINC ADDITIONS FOR GOLD AND SATURATION OF LEAD IN A 30-TON KETTLE

Up to 0.10 oz. Au per ton, 250 lb. zinc.	0.50–0.70 oz. Au per ton, 400 lb. zinc.
0.10–0.30 oz. Au per ton, 300 lb. zinc.	0.70–0.90 oz. Au per ton, 450 lb. zinc.
0.30–0.50 oz. Au per ton, 350 lb. zinc.	etc. etc.

30-ton kettle. The 30 tons lead, dissolving 0.6 per cent. Zn, will require 360 lb. zinc, whereas 0.30 oz. Au per ton are extracted by an addition of only 300 lb. spelter.

When the gold crust has been skimmed off the kettle, 500 lb. of zinc are added for bullion assaying from 150 to 250 oz. Ag per ton. With bullion running as high as 300 or 400 oz., 550 lb. are given. After removing the first silver crust the kettle will assay generally from 30 to 40 oz. per ton. The second and final addition of silver-zinc, varying from 400 to 600 lb., will reduce the Ag-contents of the kettle down to a trace, even if it has been as high as 70 oz. Bullion running low in Au and Ag, say from 0.05 to 0.10 oz. Au and from 50 to 125 oz. Ag per ton, receives one gold-zinc and one silver-zinc, the resulting lead running less than 0.2 oz. Ag per ton and generally a trace.

The most prominent plant making gold crust is that of Port Pirie, N. S. W.,[2] where the Au is removed from a kettle assaying 70 oz. Ag and 0.3 Au per ton, and the crust treated separately by a somewhat complicated procedure necessitated by local conditions.

246. Silver-Zinc.—Refineries of the United States have prepared tables giving the amounts of zinc required for desilverizing the charges of lead bullion by means of two additions, the first to remove the Au, Cu, and most of the Ag, the second to collect the rest of the Ag. A few examples are given below.

[1] Data by Blakemore, *Tr. Austral. Inst. Min. Eng.*, 1898, V, 232.

[2] Blakemore, *Tr. Austral. Inst. Min. Eng.*, 1898, V, 221.
Delprat, *op. cit.*, 1907, XII, 23.
Bayly, *ibid.*, 79.

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Example A.—This plant desilverizes in 80-ton kettles. A kettle receives 65 tons softened bullion, to be desilverized by 35 tons second crust (from a preceding charge in the form of skim blocks) and the required new zinc.

In Table 130 are given the necessary pounds of new zinc to be added. The kettle is skimmed hot (515° C.); then are taken the hot sample (gum drop) and the cold sample for assaying. The hot sample is poured into a cold conical mold as soon as taken from the kettle; the cold sample is obtained by allowing the hot ladle-content to cool in the ladle until all the crust has risen to the surface, and the lead underneath shows a clean lustrous bluish surface when the floating crust has been pushed aside, and then pouring the lead near its freezing temperature into a similar cold mold. The hot and cold samples are assayed for Ag and the amounts of new zinc to be added taken from Table 131. The kettle is cooled only after the second zincking, when the desilverized lead in the kettle ought to contain not over 0.15 Ag per ton.

EXAMPLE A

TABLE 130.—FIRST ZINC

80-ton kettle: (65 tons of lead bullion; 35 tons skim blocks from previous charge and new zinc)

SOFTENING-FURNACE CHARGE

Assay: oz. Ag (Au) per ton	Pounds new zinc to be added
100	530
110	585
120	635
140	745
160	855
180	960
200	1060
220	1170
240	1270
260	1380
280	1490
300	1600

Example B.—This plant has 60-ton kettles which, however, receive 62 tons lead bullion. The zinc schedules, given in Tables 132 and 133, are based on this amount. The crust of the first zincking is pressed hot; that of the second, cold. The crust of the second zincking is added to the next following kettle-charge to replace part of the fresh zinc needed. In order to determine how much fresh zinc the second crust is likely to replace, a cold sample is taken from the kettle which furnished the crust after the greater part of the Ag had been removed by the first zincking, and assayed. This explains the captions 10, 15, 18, ... 60, 70 in Table 132.

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EXAMPLE A

TABLE 131.—SECOND ZINC (80-ton kettle)

Hot-sample (gum drop) assay	Cold-sample assay			
Oz. Ag per ton	Oz. Ag per ton			
	1	2	3	4
	Pounds zinc to be added			
90 (or less)	1450	1500	1550	1600
95	1475	1525	1575	1625
100	1500	1550	1600	1650
105	1525	1575	1625	1675
110	1550	1600	1650	1700
If over 110 oz. per ton, take off another press after first skim, press hot (960° F. = 515.6° C.); cool down only after second zincking.				

EXAMPLE B

TABLE 132.—FIRST ZINC (62-ton kettle)

Softening-furnace assay, oz. Ag (Au) per ton	Assay oz. per ton of cold sample of preceding kettle after first zincking											
	10	15	18	20	25	30	35	40	45	50	60	70
	Pounds zinc to be added											
150	1010	950	920	900	870	840	820	800	770	770	750	750
160	1060	1000	970	950	920	890	870	850	820	820	800	800
170	1100	1040	1010	990	960	930	910	890	860	860	840	840
180	1130	1070	1040	1020	990	960	940	920	890	890	870	870
190	1150	1090	1060	1040	1010	980	960	940	910	910	890	890
200	1160	1100	1070	1050	1020	990	970	950	920	920	900	900
210	1160	1100	1070	1050	1020	990	970	950	920	920	900	900
220	1180	1120	1090	1070	1040	1010	990	970	940	940	920	920
230	1200	1140	1110	1090	1060	1030	1010	990	960	960	940	940
240	1230	1170	1140	1120	1090	1060	1040	1020	990	990	970	970
250	1260	1200	1170	1150	1120	1090	1070	1050	1020	1020	1000	1000
260	1290	1230	1200	1180	1150	1120	1100	1080	1050	1050	1030	1030
270	1330	1270	1240	1220	1190	1160	1140	1120	1090	1090	1070	1070
280	1380	1320	1290	1270	1240	1210	1190	1170	1140	1140	1120	1120
290	1430	1370	1340	1320	1290	1260	1240	1220	1190	1190	1170	1170
300	1480	1420	1390	1370	1340	1310	1290	1270	1240	1240	1220	1220
310	1530	1470	1440	1420	1390	1360	1340	1320	1290	1290	1270	1270
320	1570	1510	1480	1460	1430	1400	1380	1360	1330	1330	1310	1310
330	1630	1570	1540	1520	1490	1460	1440	1420	1390	1390	1370	1370
340	1690	1630	1600	1580	1550	1520	1500	1480	1450	1450	1430	1430
350	1750	1690	1660	1640	1610	1580	1560	1540	1510	1510	1490	1490
360	1810	1750	1720	1700	1670	1640	1620	1600	1570	1570	1550	1550
370	1870	1810	1780	1760	1730	1700	1680	1660	1630	1630	1610	1610
380	1930	1870	1840	1820	1790	1760	1740	1720	1690	1690	1670	1670
390	1990	1930	1900	1880	1850	1820	1800	1780	1750	1750	1730	1730
400	2030	1970	1940	1920	1890	1860	1840	1820	1790	1790	1770	1770

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EXAMPLE B

TABLE 133.—SECOND ZINC (62-ton kettle)

Assay cold sample, oz. Ag per ton	Pounds zinc to be added	Assay cold sample, oz. Ag per ton	Pounds zinc to be added
10	1200	35	1500
15	1300	40	1550
18	1350	45	1600
20	1375	50	1650
25	1400	60	1700
30	1450	70	1800

Example C.—This plant has 60- and 100-ton desilverizing kettles and works in the same manner as do plants *A* and *B*. In Table 134 are given the amounts of fresh zinc needed by a 100- and a 60-ton kettle in addition to those obtained from the second crusts of the preceding kettle-charges with softening-furnace lead bullions varying in their Ag-contents. Tables 135 and 136 give the amounts of zinc needed to completely desilverize the kettle-contents; the amounts are governed by the hot- and cold-sample assays as was the case in Table 131.

EXAMPLE C

TABLE 134.—FIRST ZINC (100- and 60-ton kettles)

Softening-furnace assay, oz. Ag (Au) per ton	Pounds zinc to be added	
	100-ton kettle	60-ton kettle
60	975	650
70	1000	675
80	1025	700
90	1050	725
100	1075	750
110	1100	775
120	1125	800
130	1150	825
140	1175	850
150	1200	875
160	1225	900

Example D.—This plant desilverizes in 60-ton kettles. Table 137 gives the amounts of fresh zinc needed with softening-furnace charges assaying from 135 to 295 oz. Ag (Au) per ton, and states that 6 lb. zinc are to be added for every ounce silver (gold) in excess of 295 oz. Ag (Au) per ton. Table 138 gives the zinc needed in the second zincking. In case a kettle is not completely desilverized by the second zincking, as happens occasionally, there are given in Table 139 the amounts of zinc needed to clean the kettle by a third zincking.

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EXAMPLE C

TABLE 135.—SECOND ZINC (60-ton kettle)

Hot-sample assay, oz. Ag per ton	Cold-sample assay, oz. Ag per ton				
	1	2	3	4	5
	Pounds zinc to be added				
60	1100	1150	1200	1250	1300
70	1150	1200	1250	1300	1350
80	1200	1250	1300	1350	1400
90	1250	1300	1350	1400	1450
100	1300	1350	1400	1450	1500

EXAMPLE C

TABLE 136.—SECOND ZINC (100-ton kettle)

Hot-sample assay, oz. Ag per ton	Cold-sample assay, oz. Ag per ton				
	1	2	3	4	5
	Pounds zinc to be added				
60	1650	1700	1750	1800	1850
70	1700	1750	1800	1850	1900
80	1750	1800	1850	1900	1950
90	1800	1850	1900	1950	2000
100	1850	1900	1950	2000	2050

EXAMPLE D

TABLE 137.—FIRST ZINC (60-ton kettle)

Softening-furnace assay, cold sample, oz. Ag (Au) per ton	Lb. zinc	Softening-furnace assay, oz.	Lb. zinc	Softening-furnace assay, oz.	Lb. zinc	Softening-furnace assay, oz.	Lb. zinc	Softening-furnace assay, oz.	Lb. zinc	Softening-furnace assay, oz.	Lb. zinc	Softening-furnace assay, oz.	Lb. zinc
135	325	162	487	185	625	208	763	231	901	254	1039	277	1177
140	355	163	493	186	631	209	769	232	907	255	1045	278	1183
141	361	164	499	187	637	210	775	233	913	256	1051	279	1189
142	367	165	505	188	643	211	781	234	919	257	1057	280	1195
143	373	166	511	189	649	212	787	235	925	258	1063	281	1201
144	379	167	517	190	655	213	793	236	931	259	1069	282	1207
145	385	168	523	191	661	214	799	237	937	260	1075	283	1213
146	391	169	529	192	667	215	805	238	943	261	1081	284	1219
147	397	170	535	193	673	216	811	239	949	262	1087	285	1225
148	403	171	541	194	679	217	817	240	955	263	1093	286	1231
149	409	172	547	195	685	218	823	241	961	264	1099	287	1237
150	415	173	553	196	691	219	829	242	967	265	1105	288	1243
151	421	174	559	197	697	220	835	243	973	266	1111	289	1249
152	427	175	565	198	703	221	841	244	979	267	1117	290	1255
153	433	176	571	199	709	222	847	245	985	268	1123	291	1261
154	439	177	577	200	715	223	853	246	991	269	1129	292	1267
155	445	178	583	201	721	224	859	247	997	270	1135	293	1273
156	451	179	589	202	727	225	865	248	1003	271	1141	294	1279
157	457	180	595	203	733	226	871	249	1009	272	1147	295	1285
158	463	181	601	204	739	227	877	250	1015	273	1153	Use 6 lb. zinc	
159	469	182	607	205	745	228	883	251	1021	274	1159	for every oz.	
160	475	183	613	206	751	229	889	252	1027	275	1165	Ag in excess	
161	481	184	619	207	757	230	895	253	1033	276	1171	of 295	

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EXAMPLE D

TABLE 138.—SECOND ZINC (60-ton kettle)

Cold-sample assay, oz. Ag per ton	Lb. zinc	Cold-sample assay, oz. Ag per ton	Lb. zinc	Cold-sample assay, oz. Ag per ton	Lb. zinc
10	1575	65	2085	80	1250
15	1680	70	2125	81	2270
20	1785	71	2135	82	2290
25	1840	72	2145	83	2320
30	1895	73	2155	84	2330
35	1925	74	2165	85	2340
40	1950	75	2175	86	2355
45	1980	76	2195	87	2365
50	2005	77	2215	88	2375
55	2025	78	2230	89	2385
60	2055	79	2240	90	2390

EXAMPLE D

TABLE 139.—THIRD ZINC (60-ton kettle)

Cold-Sample assay, oz. Ag per ton	Lb. zinc
0.4	275
1.0	330
2.0	440
3.0	550
4.0	660
5.0	770
6.0	880
7.0	920

247. Desilverizing Kettles.—The kettles used for desilverization are usually spherical; sometimes they are ellipsoidal. They are from 3 ft. to 3 ft. 6 in. deep, their diameters varying with the required capacity. The early vessels used were the 15-ton kettles in the Pattinson process in which the labor involved demanded a kettle of not greater depth than 3 ft.; stirring in of zinc by hand required a kettle of about the same depth; emptying a kettle by means of the Steitz siphon (see below) limited the depth of a kettle to 3 ft. 4 in., as air at 760 mm. pressure will not balance a higher column of lead. With the invention of the Howard stirrer (Figs. 571–573) and the Miller centrifugal pumps (Figs. 563–568), the original limit of 40 in. to the depth has been abolished so that one may expect deeper kettles.

Stirring in zinc by hand restricted the capacity of a kettle to perhaps 40 tons lead; with the advent of the Howard stirrer, kettles were increased in size to hold 60 and even 100 tons softened lead bullion.

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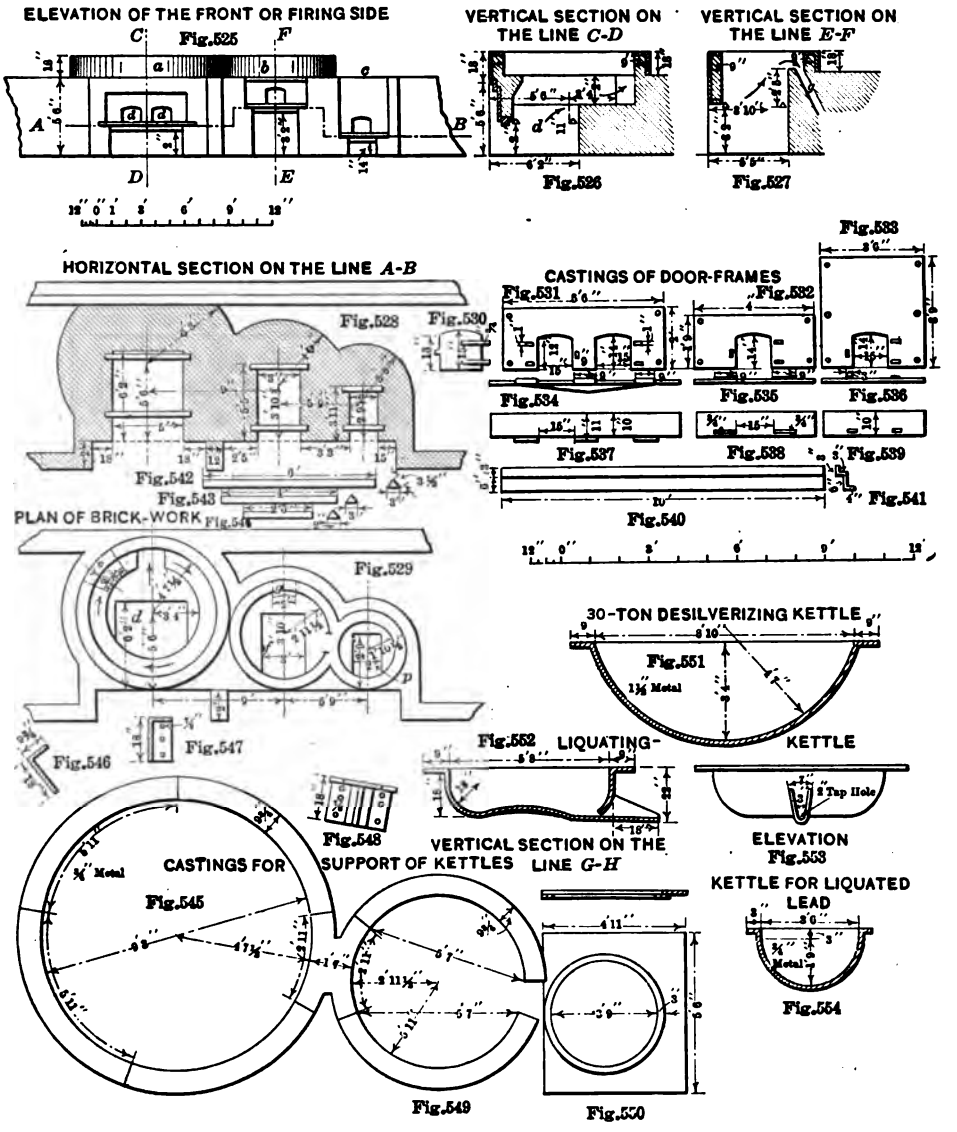
Kettles are usually of cast iron which is dense, strong, and tough; they are best cast bottom down.[1] Thirty-ton kettles are 1½ to 2 in. thick throughout; they have been made 2½ in. thick at the bottom tapering to 1½ to 2 in. at the



FIGS. 525 to 554.—Desilverizing and liquating kettles, National Smelting and Refining Co., South Chicago, Ill.

rim. Uniformity in thickness is more common. Sixty- and 100-ton kettles are heavier. A cast-iron kettle lasts from 1 to 1½ years when in continuous use.

Steel kettles have not found favor in the United States; they are in use in

[1] Lane, *Eng. Min. J.*, 1912, XCIII, 321.

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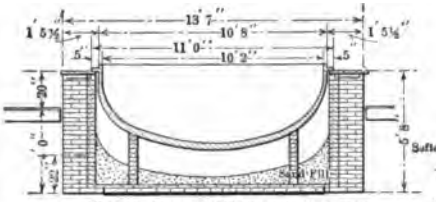
some European works in which the lead is refined by steam in the same kettle in which it has been desilverized, and give satisfaction.

Figs. 525–554 give the details of the 30-ton desilverizing kettles and the liquating kettles formerly used at the National Works, South Chicago, Ill. A kettle is usually suspended by its rim, which rests on a circular cast-iron ring covering the top and sides of a brick wall. Figs. 545, 549, 550 show three supporting rings for the desilverizing kettle, the liquating kettle, and the liquated-lead kettle. The casting (Fig. 545) consists of four separate pieces,



VERTICAL SECTION OF DESILVERIZING KETTLE

FIG. 555

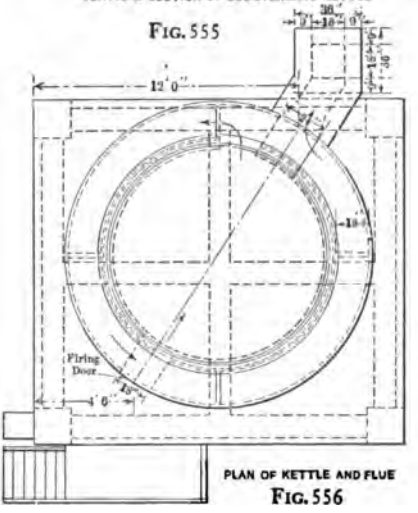
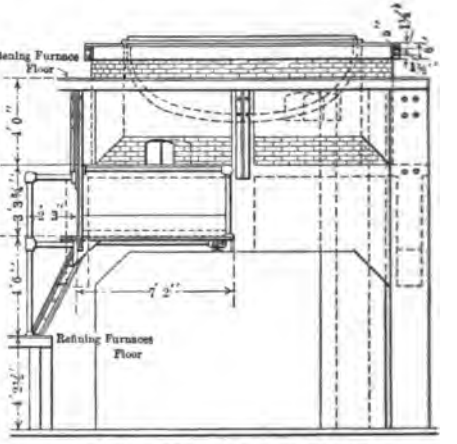


FIG. 556



ELEVATION OF KETTLE, FLUE AND PLATFORM

FIG. 557

FIGS. 555 to 557.—Sixty-ton circular desilverizing kettle, coal-fired.

shown in section by Fig. 546. They are fastened together by bolts passing through flanges, as seen in Fig. 547. The casting rests on the working-platform of the kettle, as shown in the front elevation (Fig. 525). It is made 9¾ in. wide, and incloses the 9-in. wall, which rises 18 in. above the main brickwork. With many kettles the support of the rim consists of a circular iron ring, covering only the top of the brickwork, which must be thicker than 9 in., if it is not to give way to the pressure of the weight of the kettle filled with lead. The side wall thus often reaches a thickness of 18 in., the iron support-ring either entirely covering it or leaving 2 or 3 in. exposed.[1] The place from which the kettles are

[1] Müller (*Berg. Hüttenm. Z.*, 1889, XLVIII, 218) gives it as his experience that a kettle lasts longer if suspended from a rib cast on the kettle at half its depth. This rib would then form the partition wall between the fireplace and the encircling flue, which simplifies the construction of the brick-work.

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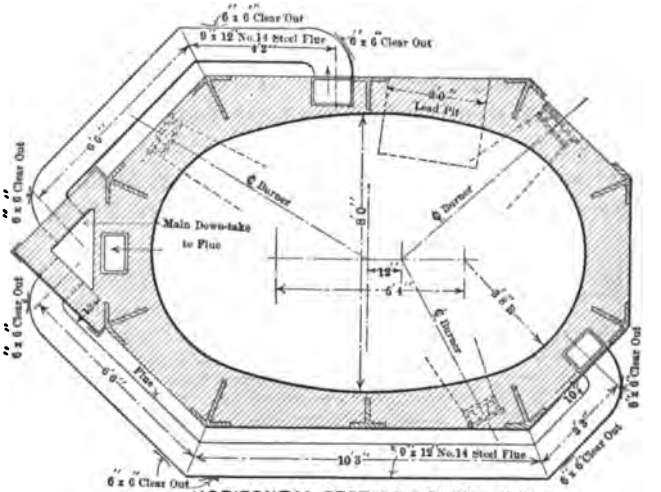
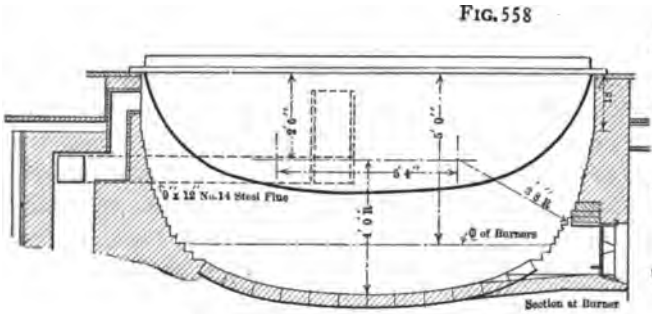
fired is seen in the front elevation (Fig. 525). The horizontal section (Fig. 528) shows the plan of the brickwork with the ash-pits of the three kettles.

Figs. 526, 527, 529 give more detail of the brickwork, and show the course of the products of combustion from the grate to the flue leading to the chimney. In the desilverizing kettle the flame goes from the fireplace, *d* (Figs. 526 and 529), first back and upward; it then passes around the kettle to the right, in a cir-


6 x 6 Clear Out
9 x 12 No. 14 Steel Flue
6 x 6 Clear Out
Lead Pit
℄ Burner
Main Down-take to Flue
℄ Burner
℄ Burner
6 x 6 Clear Out
6 x 6 Clear Out
6 x 6 Clear Out
9 x 12 No. 14 Steel Flue
6 x 6 Clear Out
6 x 6 Clear Out

HORIZONTAL SECTION BELOW FLOOR

FIG. 558



VERTICAL LONGITUDINAL SECTION

FIG. 559

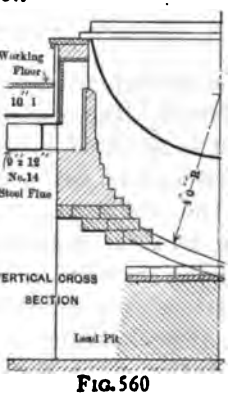


FIG. 560

FIGS. 558 to 560.—Eighty-ton oval desilverizing kettle, oil fired.

cular flue (as indicated by the arrows), and leaves this at *e*, entering a vertical flue leading to the main chimney. In the liquating kettle the products of combustion go from the fireplace, *f* (Figs. 527 and 529), after passing under the kettle, straight into the flue, *g*. The gases from beneath the liquated-lead kettle, *h* (Fig. 529), go to the left, and join those of the liquating kettle.

In Figs. 555–557 are shown the elevation, plan, and vertical section of a 60-ton coal-fired desilverizing kettle, 10 ft. 2 in. in diameter and 3 ft. 10 in. deep, 4½ in. thick at bottom tapering to 2 in. at top. The 18-in. side-walls, of red

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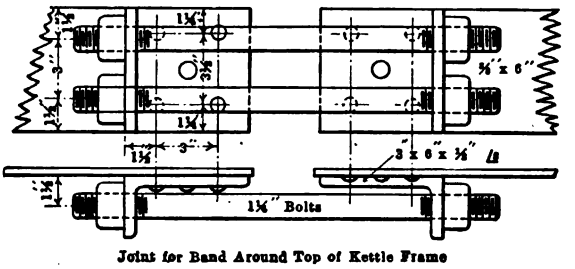
brick with a half-course of low-grade fire-brick, carry a heavy cast-iron plate which supports the rim of the kettle. Beneath the kettle are a sand-fill, shaped to the form of the kettle, and two flue-walls. The firing-door is 18 in. wide; the flame travels backward between the two flue-walls, passes to the left (Fig. 556), encircles the upper part of the kettle, and then enters the flue. The working-platform is on the same level as the floor of the softening furnace; the vertical distance between the floors of the softening and refining furnaces is 11 ft. 9¾ in., a stair-case connecting the two.

In Figs. 558–560 are given a horizontal and two vertical sections of an oval oil-fired 80-ton desilverizing kettle, 8 by 13 ft. and 3 ft. 6 in. deep, and in Fig. 561 the detail of the band which encloses the kettle-frame. The horizontal section, Fig. 558, shows a frame-work of heavy T-shaped castings, which are anchored in the ground and tied at the tops with a soft-steel band represented in Fig. 561. In the frame is erected the brick wall which carries the kettle. The wall



Joint for Band Around Top of Kettle Frame

FIG. 561.—Detail for band of eighty-ton kettle.

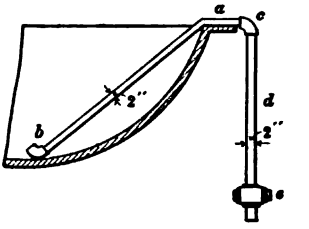


FIG. 562.—Steitz lead siphon.

is covered by a heavy cast-iron plate with flange which encloses the tops of the frame. The heating chamber, or lead-pit, has the form of an inverted arch; it is provided with an entrance 3 ft. wide. The kettle is heated with three oil-burners, two placed near one end of the kettle and the third near the other. The products of combustion are withdrawn through three ports; two end in 9- by 12-in. flues of No. 14 steel plate which deliver the gases to the main down-take; the third is close to the latter, which passes into a main flue under-ground, connected with a chimney.

Desilverizing kettles were formerly emptied by a discharge-pipe cast in the bottom of the kettle and running out through the brickwork. It was closed either by a slide-valve on the outside (similar to the lead discharge of the Luce-Rozan crystallizer), or by a clamp and thumbscrew on the inside. In 1877, A. Steitz invented[1] his siphon which quickly replaced the tapping spout, and until recently was preferable to any other means for emptying a kettle. A common form of it is shown in Fig. 562. It consists of a piece of gas-pipe, *a*, from 2 to 2½ in. in diameter, bent so as to reach from the rim of the kettle to the bottom. Here it has an elbow, *b*, screwed to it to prevent the lead column from breaking. To the other end is attached, also by an elbow, *c*, the vertical section-arm, *d*, having a cast-iron stop-cock, *e*, near the lower end. The siphon discharges the

[1] U. S. Patent, No. 196997, Nov. 13, 1877.

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lead into a cast-iron trough of ⅜-in. iron or a channel-iron or an I-beam which carries it into the refining furnace.

Since the advent of the Miller centrifugal lead pump the Steitz siphon is being replaced by the former wherever conditions permit this exchange, as it empties a kettle very quickly, *e.g.*, 60 tons lead in from 12 to 15 min. through a 2-in. pipe, the shaft driven by a 10-h.p. motor making 800 r.p.m. This pump was installed in 1905 at Trail, B.C.;[1] it replaced the Roesing pump[2] which had been in operation, and had given much trouble on account of repairs and the formation of large amounts of dross. The pump is an iron

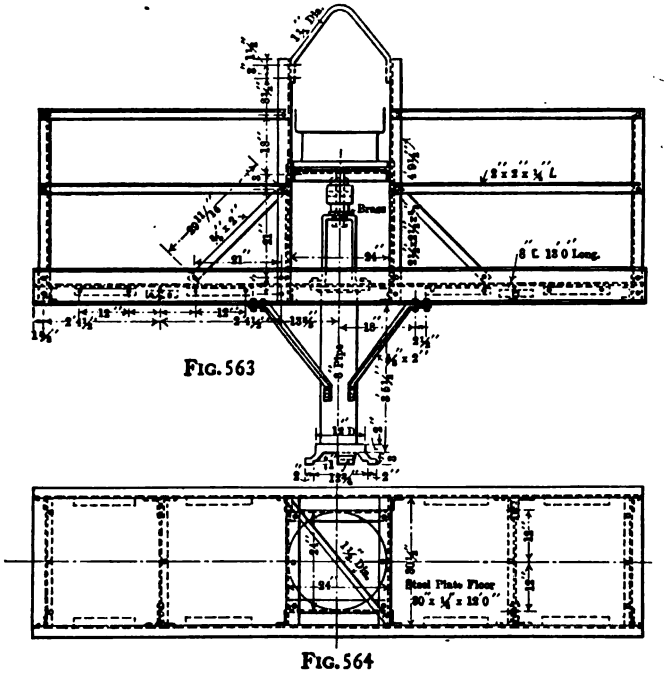

FIG. 564

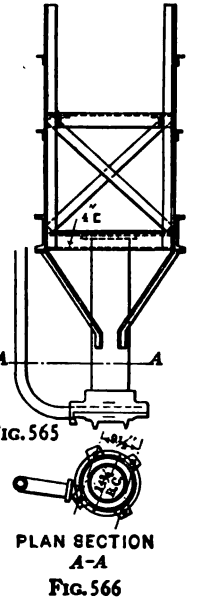

FIG. 565

PLAN SECTION
A-A

FIG. 566

FIGS. 563 to 566.—Miller centrifugal lead pump and support.

vertical submerged centrifugal pump, No. 3,[3] which has a suction of 2.5 in., and a discharge of 2.5 in. in diam., is rated at 150 gal. water per min., and costs $55. The pump is run at from 400 to 500 r.p.m. depending upon the amount it is desired to raise in a given time. At Grasselli, Ind., a similar pump 8 in. in diameter with a 2-in. discharge is in operation for molding bars. In order to have an even flow of lead, the vertical stand-pipe in which the lead rises, has a special discharge opening in the side. By keeping the lead in the pipe above the discharge opening, there is an even flow through the molding pipe which fills the molds uniformly and does away with splashing or slopping.

[1] Miller, *Eng. Min. J.*, 1913, XCV, 386.

[2] *Eng. Min. J.*, 1885, XL, 366; *Berg. Hüttenm. Z.*, 1889, XLVIII, 262; *Zt. Berg. Hütt. Sal. W. i. Pr.*, 1893, XLI, 284.

[3] Rumsey and Co., Ltd., Seneca Falls, N. Y.

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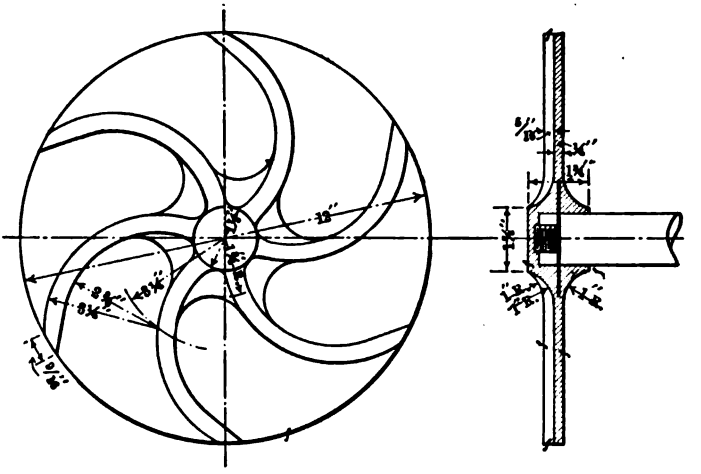
A Miller pump with its frame-work is shown in Figs. 563–566, and the impeller and shaft in Figs. 567 and 568. The frame (Figs. 563 and 564), 12 ft. long by 5 ft. 2½ in. and 4 ft. 6 in. high, is handled from an overhead crane. It carries in the center a division, 2 ft. square, which holds an 8-in. pipe to which is attached a 12-in. casing with foot-piece and a 2-in. delivery-pipe (Figs. 565 and 566). The 12-in. impeller (Figs. 567 and 568) is attached to a vertical shaft, 1⅞ in. in diameter, which is driven by a vertical 10-h.p. electric motor (not shown) at the rate of 800 r.p.m. When in operation, the frame rests upon the rim of the kettle; a steel-plate platform, 30 in. wide, on either side of the central division provides a standing place for the attendant.



FIGS. 567 and 568.—Impeller of Miller lead pump.

248. Liquating Apparatus.—In connection with the desilverizing kettle must be discussed the apparatus required for liquating the zinc crusts. Three have been in use. The last has about replaced the other two in use.

The first is shown in Figs. 552–554, page 537. It consists of a shallow kettle placed on the same level with the desilverizing kettle and close to it. The bottom is convex in the center, in order that the discharging spout may be shorter than would be possible if the kettle had the usual spherical form. Formerly a perforated cast-iron disk (an old skimmer) was placed inside the kettle, over the opening into the spout, to prevent particles of crust from passing off with the liquid lead; this has been given up at some works, as the perforations easily become clogged, and when open do not prevent fine particles from being carried off by the liquated lead. This runs from the spout into a small spherical kettle, whence it is bailed out after it has been skimmed.

The drawings given in Figs. 497 and 498 show for every desilverizing kettle one liquating kettle with its liquated-lead kettle. This presupposes that no distinction is made between gold crusts and silver crusts. Where the crusts are kept separate, the desilverizing kettle will have a liquating kettle on either side, one for the gold crust, the other for the silver crust; the liquated crusts as well

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as the liquated lead from the two small spherical kettles being kept separate. With a plant where the two crusts are kept separate, the distance of 32 ft. between the centers of the two desilverizing kettles, as shown in the general plan (Fig. 498), will be too small; it will have to be enlarged sufficiently to leave a passageway between the two small liquated-lead kettles.

The second apparatus is a reverberatory furnace placed on the floor of the desilverizing kettles. It may be built as follows: The hearth is a cast-iron plate, 10 by 5 ft., slightly trough-shaped, and having a rim 4 in. high along the sides. To the lower end a spout is attached, which discharges the liquated lead outside of the furnace into a small liquated-lead kettle of the same form as the one described, the rim of which is on a level with the working floor. The plate has an inclination of 3 in. It lies on a bed of brasque, which is tamped into a wrought-iron pan supported by transverse rails resting on two longitudinal walls. The furnace has two working-doors on one side. The writer prefers the reverberatory furnace to the liquating kettle, as he obtained from it in a shorter time than from the kettle a dryer crust, and thus a richer bullion, without driving some of the silver from the crust back into the liquated lead, as often happens with the kettle. The reason for the better result in the reverberatory furnace is probably that the lead can be gradually eliminated at a slowly increasing temperature in a partially reducing atmosphere, which prevents the oxidation that always takes place in a liquating kettle on raising the temperature to the required degree, even if it be covered with a sheet-iron plate. Supposing, however, the liquation to be equally good with the two apparatus, there remains this advantage for the reverberatory, that the liquation of the silver crusts of several systems can be performed continuously in one furnace, which, being separated from the desilverizing kettles, does not disturb the work there, and which collects all the rich crusts into one place, whence they are delivered through a chute into the bins of the retort-room. With a small plant a liquating reverberatory furnace of any reasonable dimensions would not have sufficient work to do to pay for the extra labor. The liquating kettle would have to be used, even if it obstructs to some extent the work in the desilverizing kettle. The preference for the reverberatory furnace is not general, as several important refining works adhere to the liquating kettle.

The third apparatus is the Howard alloy press,[1] shown in Figs. 569 and 570. It consists of a steel frame, usually suspended from an overhead crane, Fig. 499, which carries near the upper end a cast-iron air- or steam-cylinder of 27-in. bore with a piston of 12-in. stroke, and at the lower a heavy perforated tilting cast-iron plate. The plate forms the bottom of a movable cast-iron ring, with flaring inner side, which is held in position by flanges gliding on a vertical member of the frame. The ring in its lower position rests on the perforated bottom-plate, and is suspended in its upper position, by means of lugs, from the plunger, which has projecting teeth on the lower side. When filled with crust, the frame is raised to elevate the bottom-plate above the surface of the lead; the plunger is forced downward to remove, with a pressure of 90 lb. per sq. in.,

[1] U. S. Patent No. 563769, July 14, 1896; later form, *Min. Ind.*, 1902, XI, 452.

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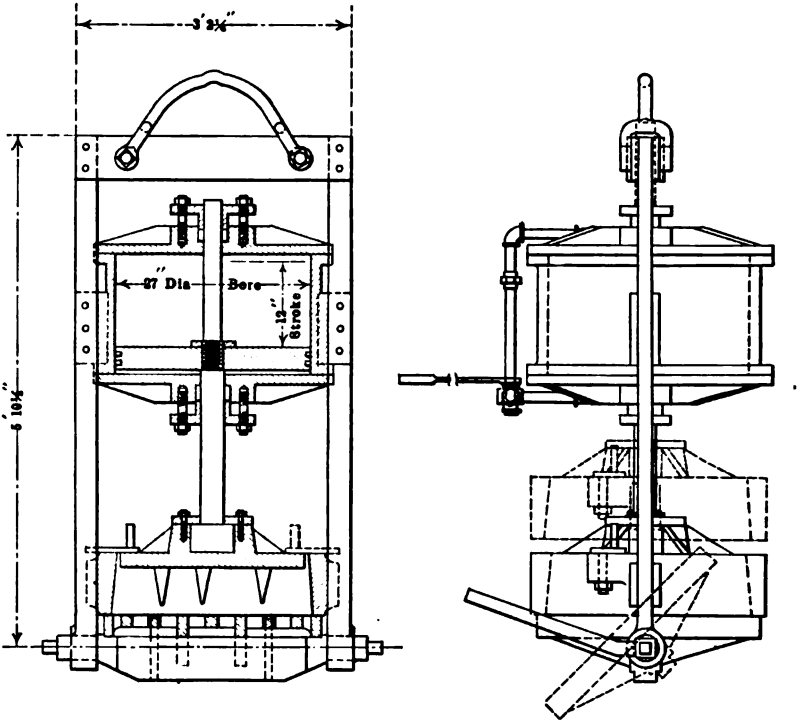
the excess lead carried by the crust, which flows back into the kettle; and the ring is then attached to the plunger. The whole apparatus is now swung to one side, the bottom-plate carrying the squeezed crust is lowered, tilted by means of a lever, and the crust dumped. The bottom is returned to its original position; the plunger carrying the ring is raised, the bottom following suit until it encloses the plunger; the ring is detached from the plunger and rests again on the perforated bottom. The apparatus is now returned to its original position over the kettle, and lowered until the ring is partly submerged in the



FIGS. 569 and 570.—Howard alloy press.

lead, when it is ready to receive fresh crust. The dumped crust is easily broken to nut-size pieces while hot, so that it may be charged into the retort. This may be done by flattening on the floor with the back of a shovel or working with a rake or hoe. Another method is to work the crust through a cast-iron box, 24 by 18 in. and 3 in. high, having a bottom ½ in. thick with openings 2 in. square.

The press is used almost everywhere for freeing rich silver crust from adhering lead; it also finds application in removing entangled lead from drosses while they are being skimmed from lead which has been melted in a kettle. The advantages of the press are, that it does away with liquating kettles and reverberatory furnaces as well as all extraneous heat, thus giving more room around the kettles and simplifying and cheapening the operation (one press

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serving a number of kettles); that, not being oxidized, the crust gives up its lead readily; and that the liquated lead flows directly into the desilverizing kettle where it belongs.

The practice of liquating zinc crusts in a spherical kettle without a lead-discharge and removing the liquated crust floating on the liquid lead with a skimmer is antiquated, if the crust is to be distilled; and justly so, as it is impossible to obtain in this way a dry crust that is satisfactory. If the temperature of the lead be raised sufficiently high to obtain a dry crust, a considerable quantity of it will be redissolved by the lead. It will rise again to the surface when the lead cools, but it will be rich in lead, and at the same time low in silver. This method is in operation with the second or final unsaturated crust which is cast into molds and goes into the next kettle-charge to take the place of part of the new zinc that is to be added.

249. Mode of Conducting the Desilverization and Results.—The lead bullion is tapped from the softening furnace, run into the desilvering kettle in about 1 hr., and is then drossed in about 20 min. It represents about 90 per cent. of the bullion charged into the softening furnace. The bullion in the kettle is heated above the melting-point of zinc (419° C.), if necessary, but usually its temperature is already higher than that of the kettle. The drossed bullion receives 10 ± tons of floor zinc, *i.e.*, the unsaturated zinc crust from a preceding second zincking which has been skimmed into molds, say 18 by 48 by 9 in. The cakes are melted in about 1 hr., then slabs of spelter for the first zincking are added, and melted, and the whole stirred mechanically for about 20 min. The rising crust is skimmed and pressed at a temperature of 480 to 510° C. in from 1 to 1¼ hr.

At some works the floor zinc is first charged into the kettle; when molten, there is added the spelter; and then only the lead bullion tapped into the kettle from the softening furnace; and the whole stirred for 20 to 30 min. Operators following this mode of procedure find or believe that the desilverization progresses more quickly and satisfactorily.

Whatever may be the sequence of charging the kettle with lead bullion, unsaturated zinc crust, and spelter, the crust and spelter have to be incorporated in the lead bullion in order to bring the zinc into intimate contact with the precious metal; and this is accomplished by stirring.

At some works, following the example of Raht,[1] there is added to the kettle, before stirring, a shovelful of salammoniac, about 1 lb. to 10 tons of lead, as it is said to counteract oxidation of crust and to assist in keeping the tools clean. Attention may be called to the use of this salt by Roessler and Edelmann,[2] in their experiments upon the Roessler-Edelmann process (§244), to free zinc crust from oxidized material.

Hand-stirring.—Formerly all zinc used to be stirred in by hand; this is still done at some works with the second zincking. In the late seventies steam was first used by Eurich at the works of the Pennsylvania Lead Works,

[1] U. S. Patent No. 826114, July 17, 1906.

[2] *Berg. Hüttenm. Z.*, 1890, XLIX, 245.

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Carnegie, Pa., and his example was followed by other plants. This method has had to give way to the Howard mechanical stirrer.

In hand-stirring, a paddle is used, consisting of a perforated disk 12 in. in diameter, riveted to a handle 6 ft. long, having a cross-piece as hand-hold. Two men standing opposite each other do the stirring. They insert the paddles vertically at the rim of the kettle, push them downward toward the center, then raise them, using the rim of the kettle as a fulcrum, and draw them, with the disks gliding on the lead, from the center toward the periphery, giving the lead a rotary motion; this they reverse every five minutes, and thus insure an intimate mixing of zinc and lead. This stirring-in, which lasts from one-half to three-quarters of an hour, is hard for the men.

Steam-stirring.—The effect of steam when introduced into lead containing zinc varies with the temperature of the lead.

1. If the temperature of the lead be below the melting-point of zinc, *i.e.*, the temperature when the kettle is skimmed, the steam will bring to the surface a zinc crust, and with it some of the silver contained in the lead.

2. If the temperature be slightly above the melting-point of zinc (stirring in time), the steam will cause a thorough mixing of zinc and lead.

3. If the temperature be between a dark-red and an incipient cherry-red, the steam will cause a scum to rise, containing about 3 per cent. Zn, which does not, however, take any silver away from the lead.[1]

4. If it be a clear cherry-red, the zinc will decompose the steam; the resulting zinc oxide (mixed with lead oxide) collects as a powder on the surface of the lead.

The steam must be absolutely dry if violent explosions are to be avoided. The condensed water is separated by a steam trap placed beneath the working-platform. To the vertical pipe coming through the platform is fastened by means of a coupling a small piece of pipe, to which is joined by two elbows (with a nipple intervening) a horizontal pipe, which reaches to the center of the kettle. An elbow connects it with the vertical pipe, that will reach 24 in. into the kettle when in place. Before the steam valve is opened, the vertical pipe is turned up, in order that the steam may first pass out into the air to warm the pipe and to expel any condensed water. The pipe is then turned down and weighted with a bar of lead to keep it in place. When the steam is turned on, the waves of lead caused by the first ascending bubbles will drive the zinc floating on the lead toward the rim of the kettle, and hardly any of it would become incorporated with the lead if it were not pushed toward the center of the kettle, to be drawn into the lead by the downward current close to the steam-pipe. The zinc crusts that rise to the surface, soon after stirring in has begun, are also pushed toward the center, that they may take up more silver. Thus the zinc and then the crusts pass down at the center, and come up again nearer the circumference of the kettle, whence they are again pushed toward the center.

The tool used for this purpose is a wooden hoe, consisting of an inch board, 12 by 18 in., into the center of which is inserted an inch lath from 8 to 10 ft. long.

[1] Roesing, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1889, xxxvii, 76, 77.

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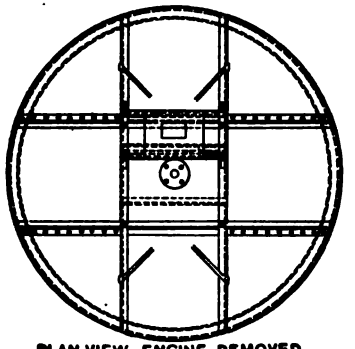
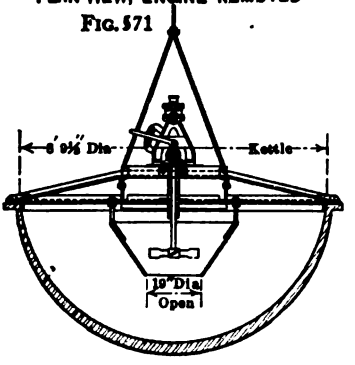
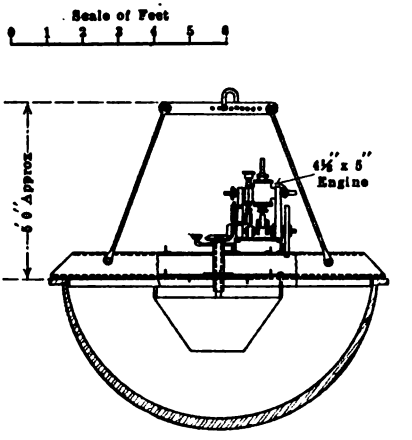
Mechanical Stirring.—This became successful only with the invention of the "Howard Zinc Stirring Machine and Cover" invented by W. H. Howard[1] and first put into operation at the refinery of the Pueblo Smelting and Refining Co., Pueblo, Colo. The mechanical stirrers of Cordurié and others[2] had became obsolete many years ago. The latest form of the Howard stirrer is given in Figs. 571–573. It consists of a steel frame-work, suspended from an overhead crane, which carries a cover, a stirrer, and a 4-by 5-in. oscillating engine. The cover, of sheet-iron strengthened by T-irons, has an angle-iron flange which



PLAN VIEW, ENGINE REMOVED
FIG. 571



SECTIONAL ELEVATION
FIG. 572



ELEVATION
FIG. 573

FIGS. 571 to 573.—Howard zinc-stirring machine and cover.

dips into the lead when the machine is in place, and seals the surface. This prevents air from passing over the metal surface while the zinc is being stirred in; and the air, being a bad conductor of heat, reduces loss of heat by radiation. The stirrer is a vertical shaft with four propeller blades which revolve in a sheet-iron cone suspended from rods to a depth that it is to be immersed in the lead. The oscillating engine rotates the propeller shaft through a double crank and a bevel gear at the rate of 110 r.p.m.

With the elliptical 80-ton kettle, shown in Figs. 558–560, two propellers with a central engine are in use.

[1] U. S. Patent No. 529617, Nov. 20, 1894.

[2] Grüner, "L'état actuel de la métallurgie du plomb," Dunod, Paris, 1868, 72.

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By the action of the propeller the metal in the inclosing cone is forced out at the bottom, being constantly replenished from the top. Thus a perfect circulation and stirring in, first of the zinc and of the zinc crust that has risen to the surface, is obtained in the short space of 10 min. with a 30-ton kettle, and in from 20 to 30 min. with a 60- or 100-ton kettle. By reversing the engine the flow of metal will be in the opposite direction. As the air is almost completely excluded, the oxidation of zinc and lead is greatly reduced.

The stirrers used at present are without cover, as this makes the apparatus cumbersome, and as the advantages expected from it have not been fully realized. It is true that some oxide is formed which collects in a small heap on the crust at center, but the amount is insignificant.

Most refineries have also given up the central cone, as with the propeller shaft running at the rate of 130 to 140 r.p.m., there is formed a vortex nearly as large as the cone. It also has been found that reversing the rotation of the shaft is unnecessary.

In outline the mode of operation is as follows: When the softened lead bullion in the kettle has been drossed and brought to required temperature, cakes of crust from the preceding second zincking are charged, and the new zinc added. The temperature of metal bath ought to be about 900° F. (=482° C.). The Howard stirrer, which has been lowered and put in place, is rotated for from 20 to 30 min.; it takes some time for the floating zinc crust from the charged cakes to become thoroughly incorporated. The first crust is now taken and pressed hot in a Howard press in from 1½ to 2 hr. Hot and cold samples are taken and assayed. The second zinc addition is made and stirred in with the Howard machine. The kettle-contents are now cooled by banking the fire under the kettle, and by spraying water, at center on the surface of the metal bath, from a ¾-in. pipe which has small perforations on the lower side and is closed at the end. The crust is taken off in from 1½ to 2 hr.

In stirring in the zinc, it is of importance that the lead have the correct temperature; if the latter is too low, the zinc will be imperfectly incorporated; if too high, the crust will run too low in precious metal. Whether this is due to the crust taking up lead, or the lead dissolving crust is not definitely known, as the behavior of Ag_2Zn_3 in this respect has not been studied. The work of Alder-Wright and Thompson,[1] Kreman and Hofmeier,[2] and Bagitsch[3] lead one to believe that the crust is dissolved in lead. Refiners stir the zinc into lead bullion which has been brought to a temperature ranging from 480 to 510° C. Large-scale tests by Newton[4] have shown that in stirring in the zinc at 705° C., over 40° C. above the freezing-point of Ag_2Zn_3, and removing from the crust the excess-lead with a Howard press the assays of the crust, or rather of the resulting retort bullion, increased with temperatures falling to about 500° C., and then decreased owing to the inability of the press to remove the excess-lead at the

[1] *Proc. Roy. Soc.*, 1890, XLVIII, 32; *Eng. Min. J.*, 1890, L, 718.
[2] *Monatshefte für Chemie.*, 1911, XXXII, 563, 597.
[3] *Compt. rend.*, 1914, CLIX, 1781.
[4] *Tr. A. I. M. E.*, 1915, LI, 786.

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prevailing low temperature of 400 to 430° C., even with a pressure of 90 lb. per sq. in.

It was formerly the custom to cool the lead after each zincking in order that the crust might separate as much as possible from the lead, and be skimmed off completely. This practice, which takes about 3 hr. for every zincking, prevails with the second crust, but has been abandoned with the first, as already shown. When the first zinc has been incorporated in the lead, the kettle is skimmed with the fire kept going on the grate. When skimmed, hot (gum drop) and cold samples are taken. The hot sample is poured from the ladle as soon as it has been taken; the cold is poured after the ladle-content has cooled sufficiently to allow *all* the crust to rise to the surface and to permit casting a sample of clean lead. The cold sample represents the condition in which the lead in the kettle would be after cooling and skimming all the crust. The zinc necessary for the second zincking is found in Tables 131, 133, 135 and 138. By taking hot and cold samples and proceeding without cooling the kettle, a considerable saving in time is attained. At present the two usual zinckings require 8 hours excluding the inflow of the lead, but including the outflow. With the older method of operating, each zincking of a 60-ton kettle lasted not less than 6, and often 8 hours.

When the first zinc-silver crust has been removed, and the second zinc has been added and stirred in, it is necessary to remove all the crust that can be made to rise to the surface. For this purpose the kettle-content has to be cooled. Usually the fire on the grate is removed or damped with slack coal, and the fire- and ash-pit doors, and the damper in the flue are thrown wide open that air may rush through the fireplace and cool the kettle.

A number of devices for hastening the cooling have been tried, but had to be given up. Thus Meyer[1] used water-cooled pipes bent to the shape of the kettle. Others have blown air on to the lead through a pipe, 4 to 5 in. in diameter, suspended vertically so as to be about 3 ft. above the bath. Water is frequently sprayed on to the lead, as shown above; and the last two blocks of second crust from the preceding charge are added to assist in the cooling; in addition steam is blown through the bath, which also assists in raising to the surface the last particles of crust.

When the lead has cooled sufficiently for the crust to adhere to the sides of the kettle, the crust is removed with a skimmer; its disk, made of ⅛-in. iron, is from 15 to 18 in. in diameter, and the handle, of 1⅛-in. gas-pipe, is 7 ft. long, having a cross-piece as hand-hold. The work is facilitated by suspending the skimmer from a hook, which acts as a lever; or the kettle-rim has a fork which acts similarly. Two men work together. One man with a wooden hoe pushes the crust toward his partner, who takes it up with the skimmer. Before discharging it into the Howard press or the mold, it is important that the skimmer be well jerked several times, in order that as much lead as possible may be drained off into the kettle, and the crust obtained become dry. Toward the end of the operation both men have to work very slowly and carefully to

[1] *Min. Sc. Press*, 1882, XLIV, No. 5; *Berg. Hüttenm. Z.*, 1882, XLI, 391.

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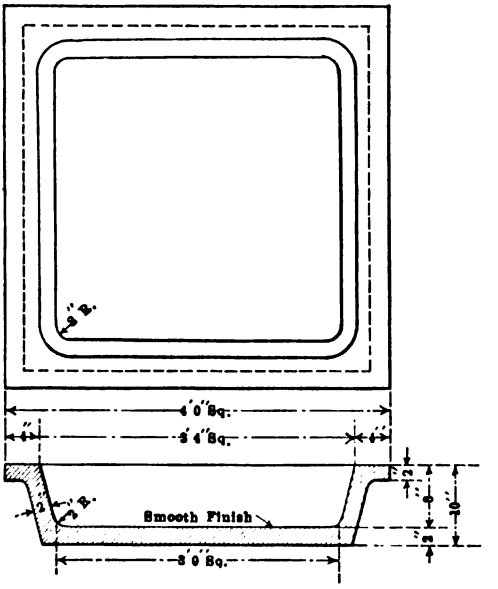
avoid pushing the crusts back again into the lead, which would retard the work. When the crusts have been removed from the surface, the alloy adhering to the side of the kettle has to be brought to the surface. This is done by scraping the kettle, first with a chisel-pointed bar and then with a wooden lath. The blade of the bar is of steel, 2 by 4 in., and the handle of 1-in. round iron. Skimming and scraping are generally repeated twice, after which no more crusts will rise to the surface. It takes from 1 to 2 hr. to perform this operation. When finished, the fire under the kettle is started, and the kettle heated up for the next zinc. The time required for heating is 1 to 2 hr.

The second crusts are usually collected in molds or discharged on a white-washed iron plate. Such a mold is shown in Figs. 574 and 575. There the crusts remain until needed, when the molds are raised and emptied by means of a crane, or one end of a plate is raised by block and tackle, and the crusts are slid into the kettle.



FIGS. 574 and 575.—Mold for zinc crust.

After each skimming, samples are taken, and ½ assay-ton samples are assayed for Ag to check the progress of the desilverization. After the last silver crust has been removed, the assay should show 0.2 oz. Ag per ton, or less if corroding lead is being made. Should it prove to be slightly higher, say 0.4 or 0.6 oz., the introduction of steam at the low temperature of the kettle after the last crust has been taken off, will be effective in causing more silver-bearing zinc crust to be given off by the lead, as stated above. If the steam is used from one-half to three-quarters of an hour, the silver-contents of the kettle will be reduced, and thereby generally an entire zincking saved.

The weights of the crusts vary considerably as do the amounts of silver they take up. Assuming a 60-ton kettle, and 150-oz. lead bullion, the first crust will weigh 5200 lb., and the partly desilverized lead will assay 40 to 60 oz. Ag per ton; the second crust will weigh 18,000 to 19,000 lb. corresponding to five cakes, and the desilverized lead will retain <0.2 Ag per ton.

The concentration of silver from lead bullion in retort bullion is as 25 : 1. One ton fresh lead bullion requires 15.3 lb. or 0.765 per cent. zinc; with the second crust added to the kettle, this figure is reduced to 14 lb. or 0.700 per cent. The older practice required 1.25 per cent. zinc. Two zinckings require 8 hr. excluding running in, but including running out of lead.

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Two men do the work on a kettle; each kettle has its fireman, who in summer may have a helper. There are required about 2 tons coal in 8 hr.; this corresponds to 40 to 60 lb. per ton of lead bullion, depending upon the capacity of the kettle.

In connection with desilverization may be mentioned an excellent custom of breaking up unwieldy old kettles that have been set aside on account of leakage or corrosion. This is done by filling the kettle with water and suspending a dynamite cartridge from a floating board so that it nearly touches the bottom. When this is exploded, the water will be thrown up into the air and the kettle broken into five or more pieces, which can be easily handled and shipped to the foundry. Another method, which has come into use, is the acetylene cutting.

250. Refining Desilverized Lead, General.—The desilverized lead retains, after the last crust has been removed, from 0.6 to 0.7 per cent. Zn, according to the temperature that prevailed at the last skimming. To remove this, and also small quantities of As and Sb that were either not entirely taken away during the softening, or that were introduced with the zinc used in desilverizing, the lead must undergo a refining process.

From the desilverizing kettle the lead is siphoned or pumped into the apparatus used for refining, which is in most American refining works a reverberatory furnace, occasionally a spherical kettle.

The siphon (Fig. 562) is heated and filled by immersing it in the kettle, the stop-cock being open. When it has attained the temperature of the lead and is filled entirely with it, the stop-cock is closed with a key, the longer arm taken out with long-handle tongs and suspended, and the shorter one held down in the lead. The stop-cock is then opened, and the lead runs out into a trough, which discharges into the refining apparatus. To keep the siphon in place it is weighted by a couple of bars of lead. If the lead column breaks when the kettle has, for example, been half-emptied, and it is not possible then to fill the siphon again in the usual way, it will be necessary to invert and fill it by ladling; for this purpose an iron funnel may be used to avoid delay. But the breaking of the lead column is a very rare accident.

The Miller lead pump has been discussed in connection with Figs. 563–566.

251. Refining Desilverized Lead in the Reverberatory Furnace.—The general construction of the reverberatory furnace used for refining is the same as that for softening. Formerly it was customary to make the refining furnace smaller than the softening furnace, in order that it might correspond to the smaller amount of lead it had to treat. At present it is more common to give the refining furnace the same dimensions as the softening furnace, only making the hearth slightly shallower. The arrangement for tapping is usually the same in both furnaces. If, however, the refined lead is to be conveyed directly from the furnace into the molds instead of being molded from a storage reverberatory furnace or a "Merchant kettle," the tap will be slightly different from that of the softening furnace.

The mode of operating is similar to that in the softening furnace. When

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the furnace is filled, the fire is urged for 3 or 4 hr. to bring the lead to a cherry-red, as a high heat is required to burn off the zinc. A bar inserted into the bath should be red hot after a few minutes. The zinc is partly volatilized and carried off with the fumes, and partly oxidized and scorified by the litharge which forms at the same time. After heating about 4 hr. the surface of the lead will be covered with a heavy litharge-like skimming. The doors are thrown open to cool the surface of the metal, the skimming is removed, and a second heat given, after which cooling and skimming are repeated. A third heat is often necessary to slag the last traces of Zn and Sb. When these are completely removed, the litharge drawn from the surface of the lead by means of a rabble should be in large, thin plates. It should have, while hot, a bright yellow color when seen in bulk, and a greenish yellow when held up to the light; no brown spots (antimony) should be visible. If these large flakes of litharge should become dark or show spots after having been exposed for some time to the air, the lead is not sufficiently refined to satisfy the requirements for corroding lead.

A bar of lead molded at the correct temperature should show on the surface fern-like crystalline forms, the branches at right angles to the main axis. If the lead is not sufficiently refined, there will form on the surface in addition to the fern-like crystals, small bunches of crystals similar to snowflakes, which disappear again when the bar solidifies completely, leaving the surface finely crystalline. A ladle filled and emptied two or three times with lead will show in the bowl the blue color of a bar of refined lead, if all the impurities have been removed. Lastly, a bar of lead will often show on the lower side of the lip a blue color, if the lead has been satisfactorily refined, although the absence of this color will not prove that the lead is of inferior quality.

In firing the refining furnace, it is essential that the temperature to be raised quickly to the necessary intensity, and kept there. If it is allowed to fall even a little, the burning off of the zinc will be greatly retarded, and with it the dependent operations.

In order to hasten the operation, steam is used. In addition to the mere mechanical effect of stirring, it acts chemically by being decomposed by the zinc. It is introduced through four or six ¾-in. pipes inserted into the lead through the doors so as to nearly touch the hearth. The steam pressure is correct when the lead is thrown upward about 1 ft. In order to obtain skimmings thoroughly melted and free from lead, it is essential to keep the temperature high; if it is too low, the skimming is likely to be powdery and filled with shots of lead. Steam, if rightly used, will not increase the amount of skimming.

The refining with steam is continued until all the zinc has been oxidized. Skimming is begun as soon as sufficient oxide has been formed, and is continued until the charge has been freed from zinc. Two men, each with helper, remove the skimming and transfer it into a perforated pot.

The time required for refining without the use of steam, ranges from 10 to 15 hr.; with five or six steam-jets, it is 6 or 8 hr. including filling and

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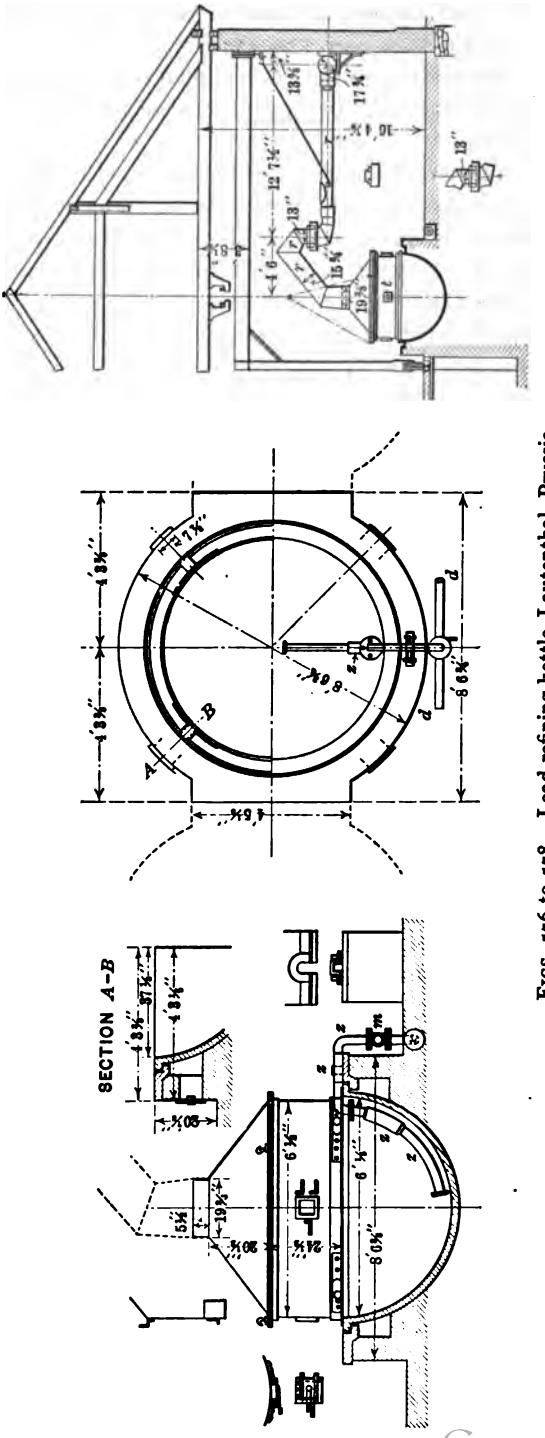
emptying of furnace. The refining skimming amounts to about 5 per cent. of the bullion charged, and contains about 90 per cent. Pb. At first it is dry, but becomes mushy toward the end of the process. The coal consumed in refining used to be about 120 lb. per ton of unsoftened bullion, corresponding to about 7 gal. of reduced oil, air being used as atomizer. At present a 100-ton furnace burns in 24 hr. about 5 tons coal and treats three charges.

252. Refining Desilverized Lead in the Kettle.— The second method of refining desilverized lead is the one invented by Cordurié, who introduced steam into the lead heated up to cherry-redness, the oxidized zinc collecting in the form of a powder on the lead. As the air cannot be excluded from the surface, and as it is also carried in by the steam, some of the lead is oxidized, and the pulverulent yellow mass floating on the surface consists of a mixture of lead and zinc oxide and finely divided shots of lead. The composition of these oxides[1] when taken from the kettles at the Lautenthal Smelting and Refining Works (Prussia) is Sb_2O_5 1.893, Fe_2O_3 0.986, ZnO 23.775, PbO 37.933, Pb 34.236 per cent.

[1] *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1890, XXXVIII, 272.



FIGS. 576 to 578.—Lead-refining kettle, Lautenthal, Prussia.

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The lead bullion at Lautenthal is desilverized without previous softening; hence the high percentage of Sb.

An analysis of the oxides taken from the refining kettle of the Pueblo Smelting and Refining Co. gave, according to Howard: ZnO 12.5, PbO 6.0, Pb 81.0 per cent.

The larger shots of lead of the oxides at Lautenthal are separated by screening, the finer ones by washing over an inclined plane. The impalpable powder, forming 15.44 per cent. of the whole, floats off and is settled in vats, dried in a reverberatory furnace, and forms a reddish yellow paint of good covering power. It consists of 60 to 67 per cent. ZnO, and 33 to 40 per cent. PbO. The residual shots of lead are smelted at intervals for a second-class lead, as they contain some antimony.

At Lautenthal[1] (Figs. 576 to 578) the cast-iron kettles (6 ft. in diameter, and 3 ft. 2 in. deep, holding 12½ metric tons of lead) are heated, after the desilverization is finished, in 4 hr., to a cherry-red; superheated steam, having a pressure of from 29 to 36 lb. per sq. in., is then introduced through a cast-iron pipe, *z*, bent to the form of the kettle, so that the steam enters at the bottom. After 2 hr. all the zinc has been oxidized. In order to decrease the loss of heat by radiation, to keep off the air, and to prevent the oxides from being lost, the kettle is covered by a movable sheet-iron cylinder, which has near the lower rim two opposite doors (4¾ in. square), and the opening for the steam-pipe. It ends in a conical hood which leads the vapor and dust through a sheet-iron pipe into the main flue, terminating in a dust chamber. The cylinder, with its hood and pipe, is suspended by a running differential pulley from a traveler. For every 100 lb. of unsoftened bullion, there are produced 4.67 lb. of paint, which is higher than the percentage of skimmings in the reverberatory furnace, as the paint forms but a small part of the total lead taken out of the kettle.

It used to be the practice to steam in two stages. At first the doors in the hood were kept closed so as to exclude the air as much as possible. When the zinc had been oxidized, the hood was removed, and the oxides were taken from the surface of the lead by means of a perforated skimmer. The hood was then put again in place, the doors were opened, and steaming was resumed with the object of removing the Sb in the form of antimony skimming, which was taken from the lead when this had cooled sufficiently. The changes in composition of the lead which take place during steaming, as given by Waldeck,[2] are shown in Table 140.

Part of the steam refining plant of the Pueblo Smelting and Refining Co., Pueblo, Colo., is shown in Figs. 579 and 580. The kettle, 8 ft. 1⅞ in. in diameter, and 3 ft. deep, is intended to treat 20 tons of desilverized lead. Its rim has two concentric rings forming ridges, between which is a groove to receive the bottom of the hood. The air can then be entirely excluded by sealing the joint with sand or oxides. The sheet-iron hood is raised and lowered by means

[1] Private notes, 1890; Illustrations from Schnabel-Louis, "Handbook of Metallurgy," Macmillan, London, 1905, 1, 550.

[2] "Streifzüge durch die Blei und Silberhütten des Oberharzes," Knapp, Halle, 1907, 57.

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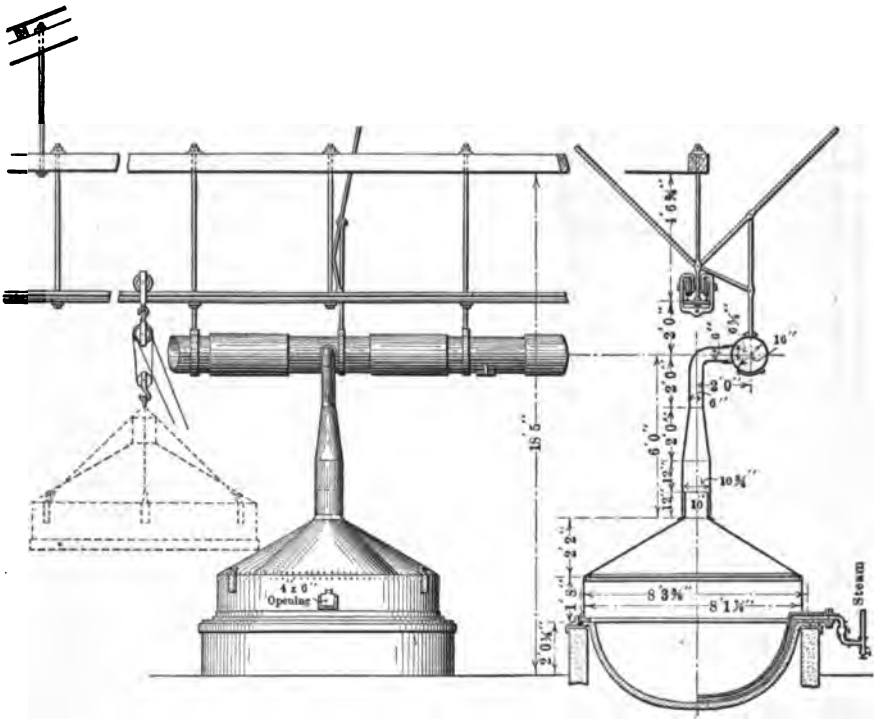
TABLE 140.—CHANGES IN DESILVERIZED LEAD BY STEAMING

Metal	Desilverized lead	Time of steaming, hr.					
		With doors of hood closed			With doors of hood open		
		0.5	1.0	1.5	0.5	1.0	1.5
Sb.....	0.6328	0.6243	0.6188	0.6134	0.3667	0.1132	0.0108
Cu.....	0.0138	0.0125	0.0123	0.0118	0.0010	0.0007	0.0005
Bi.....	0.0013	0.0013	0.0014	0.0016	0.0016	0.0017	0.0018
Zn.....	0.7010	0.0533	0.0086	0.0010	0.0008	0.0005	0.0003
Ag.....	0.0004	0.0004	0.0004	0.0004	0.0004	0.0004	0.0004
Fe.....	0.0017	0.0018	0.0018	0.0019	0.0022	0.0023	0.0025



FIGS. 579 and 580.—Lead-refining kettle, Pueblo, Colo.

of a differential pulley suspended from a traveler, and joined by a vertical pipe to a suspended horizontal main leading into the dust chamber. The dimensions of these pipes are smaller than those of the pipes at Lautenthal, because the former serve only two kettles, whereas the latter are intended for a larger number. It takes 3 hr. to heat up the kettle, and 3 hr. to steam it, about 1700 lb. of oxides being produced.

If the kettles were made larger, there would be danger of the bottom becom-

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ing weak in heating up the lead to the required temperature, and yielding to the pressure in steaming.

Instead of using steam, Savelsberg introduced into the pipe a fine spray of water under a pressure of 40 lb. per sq. in. which is converted into steam by the heat of the lead. The valve of the water-main and the pipe leading into the kettle are connected by a copper coil which takes up the shocks caused occasionally by the admission of an excess of water which is not converted into steam.

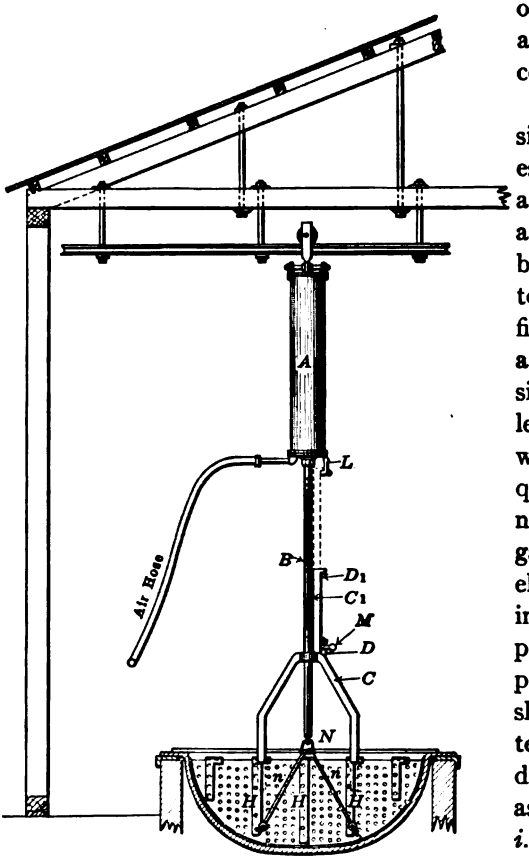

FIG. 581.—Howard skimmer.

For the success of freeing desilverized lead from zinc it is essential that the lead be kept at a cherry-red, as then the zinc, and incidentally some lead, will be quickly oxidized, and float on top of the lead in the form of a fine powder readily removed with a skimmer. If the temperature sinks, the time of steaming is lengthened, and the oxide formed will be mushy and retain a large quantity of shots of lead disseminated through it. Kuhlemann[1] gave the following facts as to the elimination of zinc: After steaming ½ hr., the lead retained 0.402 per cent. Zn, after 1½ hr. 0.116 per cent., the zincky lead still showing its characteristic of flattening out readily; when this had disappeared and the oxides had assumed the form of a dry powder, *i.e.*, after 2 hr., the lead retained 0.0004 per cent. Zn. More complete data have been given in Table 140. The operation is finished when the oxides are powdery, and a sample of lead taken up in a ladle forms clean golden yellow litharge whenever its surface is exposed to the action of the air.

The oxides are usually removed with a perforated skimmer, which is a lengthy and hot piece of work. The operation is very much simplified by the use of the Howard skimmer,[2] shown in Fig. 581. This was originally constructed to remove the zinc crusts from desilverizing kettles.[3] It has been

[1] Private notes from the Lautenthal "Record of Analyses," 1879.

[2] U. S. Patent No. 502, 122, July 25, 1893.

[3] *Min. Ind.*, 1893, II, 440.

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replaced in most instances by the Howard press, and is now used for oxides. It consists essentially of two parts: a yoke, *C*, with slotted extension, *C'*, carrying a pair of hinged skimmer-leaves, *H*, of perforated boiler iron, and a suspended air-cylinder, *A*, with piston-rod, *B*, carrying a link, *N*, with two pairs of rods, *n*, one for each of the leaves. The piston-rod carries a pin, *D*, traveling in the slot of the extension, *C'*, and the extension has a lock, *M*, to make a rigid connection between the yoke and the piston-rod; the air cylinder has on the lower side the air inlet and opposite to it a latch, *L*, to secure the projection of *C'*.

Supposing the kettle to be ready to be skimmed, and the skimmer to be immersed in the lead, as shown in the figure, compressed air will be turned on through the hose. This causes the piston-rod to rise and the leaves to unfold. The yoke remains stationary, and extending into the lead prevents the leaves from turning. When the piston-rod has opened the leaves, pin, *D*, will have moved up in slot, *C'*, to position *D'*, whereupon it engages with the yoke extension and pulls up the skimmer with its load until the projection of *C'* is latched by *L*. It is now pushed to one side, where the oxides are to be unloaded. For this purpose some air is let out of the cylinder, when the leaves will drop, the yoke remaining latched by *L*. When the skimmer is to be used again, it is brought over the kettle, the yoke locked with *M* to the piston, latch, *L*, raised, and air let out again, whereupon the closed skimmer will sink into the lead.

The use of compressed air makes the work very easy. If sufficient pressure cannot be had, the skimmer is worked by hand, substituting for the air cylinder a combination of rack and other gearing.

The great drawback of steaming zinc-bearing lead is the wear and tear on kettles. According to Schmieder,[1] at Tarnowitz, where the lead is very free from antimony, cast-iron kettles last only for 20 charges, but steel kettles are good for 90 charges. The life of cast-iron refining kettles varies greatly; extreme figures are 30 days and 120 days. Of kettles from the same foundry, cast under apparently the same conditions, one within the writer's experience will last only a month, while another lasts over a year. The life of a kettle is greatly prolonged, if it is used exclusively for dezincking lead, and has not to serve as is the case in most European works, both for desilverizing and refining.

As by the use of steam a considerable amount of air is carried into the lead and causes oxidation, Rœssler[2] tried to replace it by different gases. With CO_2, the lead being heated to 700° C., in a short time all the zinc was converted to white ZnO, and could be skimmed off from the lead. As CO_2 cannot be easily obtained pure to be used for such a purpose, he tried a mixture of CO_2 and N, obtained by passing air through a cylinder filled with glowing coal. The result was a gray powder, in which some of the zinc was present as metal in a finely divided state. By the use of a mixture of CO and N, drawn from a gas-producer, a powder of a darker gray was obtained, containing up to 75 per cent. of its zinc as metal in a finely divided state, the rest being oxidized by the CO_2 present. The refined lead was entirely freed from Zn, if the temperature

[1] *Berg. Hüttenm. Z.*, 1887, XLVI, 377.
[2] *Op. cit.*, 1890, XLIX, 248.

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was kept above 700° C.; otherwise a zinc crust formed on top of the lead. In working on a large scale with gas containing from 15 to 20 per cent. of CO_2, too many difficulties were encountered, according to Schnabel,[1] to make the process a practical success.

One process of refining desilverized zinc-bearing lead may be mentioned, as it has found favor in some European works. It is the process of F. Roderbourg[2] in which a mixture of $ZnCl_2$ and PbO is stirred into the kettle holding the lead. The action of this mixture is said to be $Zn+PbO+ZnCl_2=Zn+PbCl_2+ZnO=ZnCl_2+ZnO+Pb$. The scum floating on the lead bath is removed, and treated with H_2O to remove the $ZnCl_2$. The leached residue is heated with NaCl and PbO, and thereby the original reagent produced. The process has been tried in the United States and dropped.

253. Molding of Refined Lead.—The molding or casting of lead was formerly done by hand, being ladled from the kettle into molds placed in front of it.

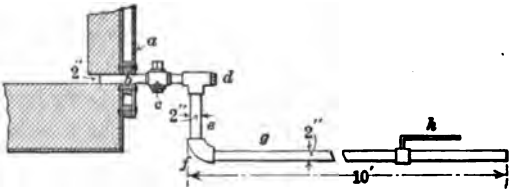

FIG. 582.—Lead-molding apparatus for refining furnace.

This method has become obsolete. At present the refined lead is run into molds either directly from the reverberatory furnace or kettle in which the zinc had been removed from the desilverized lead, or, and this is more common, the refined lead is tapped into a storage reverberatory furnace or a market kettle from which it is run into molds.

In molding from the *Refining Furnace*, a piece of 2-in. pipe, *b*, provided with a stop-cock, a T-joint and a plug, *d*, Fig. 582, is screwed into a cast-iron collar, *a*, fitted into the steel water-jacket at the tap-hole. To the vertical end of the T is attached nipple, *e*, with elbow, *f*, at the lower end, into which is screwed a long pipe, *g* (from 7 to 10 ft.), which can be moved horizontally by arm, *h*, while it discharges the lead into molds placed in a half or three-quarters, the center of which lies beneath the nipple. The molds commonly used now (Figs. 583–585) differ from the ordinary blast furnace molds in that one end rests on two wheels, *a*, and the other has a leg, *b*. The lip of the mold above the leg has a hole, *c*. By passing a hook through it and tilting the mold, it is run away, the bar tipped out, and the mold then quickly returned to its former place. At some works the molds are made large enough to hold three bars of lead. The lip then has, instead of the hole, *c*, a rectangular socket running horizontally, into which is inserted a slightly bent iron handle to move and tilt the mold.

[1] Schnabel-Louis, "Handbook of Metallurgy," I, 547.

[2] German Patent No. 241483; *Metallurgie*, 1912, IX, 239.

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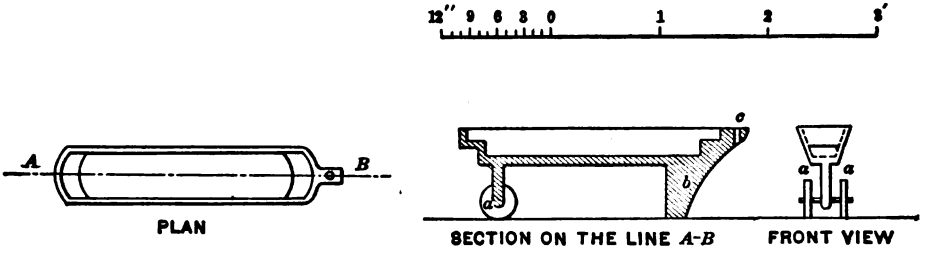
When the furnace is to be emptied, a charcoal fire is started under the stop-cock, and the horizontal pipe immersed in the lead of the refining furnace to be warmed. It is then screwed into the elbow, the stop-cock is opened, and the lead run into the first mold of the half- or three-quarters circle. This warming of the pipe is, however, not necessary. If the stop-cock be opened entirely, the first lead arriving at the end of the horizontal pipe will still be liquid. After that, the cock will have to be slightly closed, as the molds would otherwise fill too quickly for the man, who has also to attend to the skimming of the surface of the bars. This he does with two thin pieces of board the width of the mold, collecting the dross between them and dropping it on the floor. Another method is to rake off the dross on to the floor with a bent piece of hoop-iron. The former method gives a cleaner bar.



FIGS. 583 to 585.—Mold for refined lead.

One mold after another is thus filled. When the lead in the first three or four molds has solidified, it is chilled with water, the pigs are trimmed with a sharp, chisel-pointed bar, and the molds run off to the wall of the lead-pit (Fig. 498), where the lead is to be dumped before weighing, and brought back empty to their places. In this way from 40 to 60 molds that form half- or three-quarters semicircles are filled one after the other.

This method of molding has the advantage that the vertical distance between the shipping level of refined lead and the receiving level of lead bullion can be less than with the methods to be discussed; further, the filling of the molds need not be continuous, as it must be with the siphon. It has, however, the great disadvantage that the furnace has to be cooled to the molding temperature of lead, which is about 425° C. The cooling, as well as the subsequent heating, takes considerable time; further, fluctuations of temperature in a furnace are always detrimental to the brickwork. It is therefore preferred to tap the refined lead into a separate furnace, be it reverberatory or kettle, and to allow the lead to cool there. The molding of lead does not interfere with the regular working of the plant; it is usually given in contract to a separate crew, which attends to the molding, and the loading into cars.

The *Storage (Molding) Reverberatory Furnace* is in common use today with plants having large units. Its capacity is the same as that of the refining reverberatory furnace. Figs. 586–588 give a side-elevation and end-elevation at bridge-wall and flue-end of a 100-ton storage reverberatory furnace

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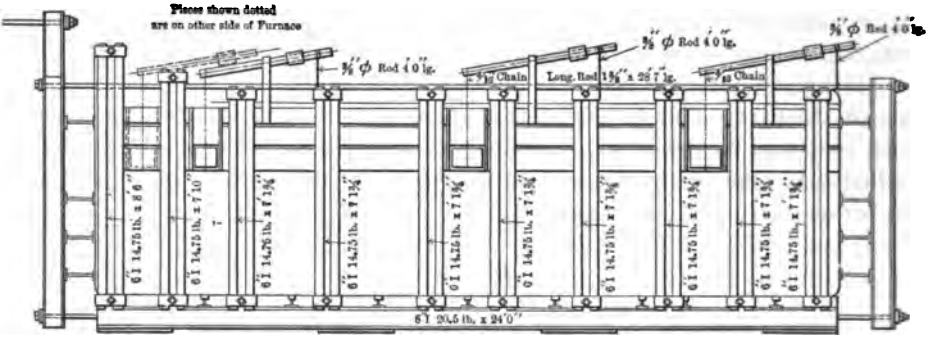
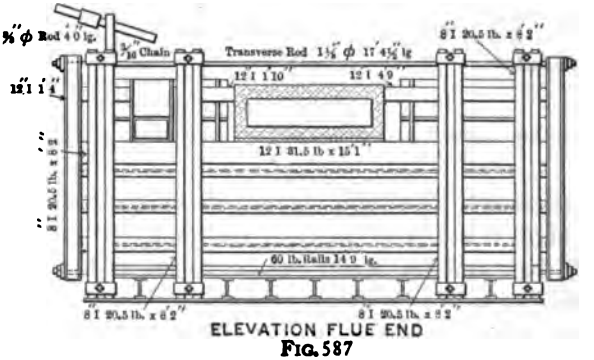
for the molding of lead; Figs. 589–591, horizontal, longitudinal, and transverse sections, and Figs. 592–594 details of the 6- by 6-ft. fireplace with step-grate.

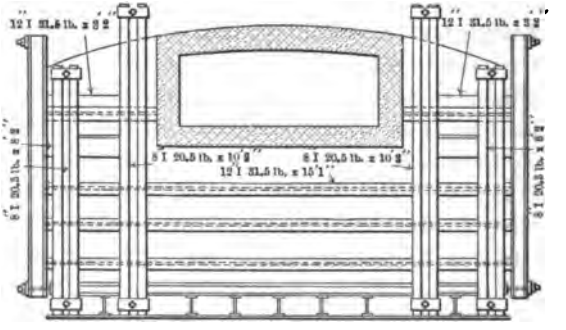
The furnace, 21 ft. 9 in. long and 12 ft. 9 in. wide, has a metal bath 3 ft. 1½


Pieces thown dotted
are on other side of Furnace
⅝" ϕ Rod 4'0" lg.
⅝" ϕ Rod 4'0" lg.
⅝" ϕ Rod 4'0" lg.
⅝" Chain
Long. Rod 1⅝" x 28'7" lg.
⅝" Chain
6 I 20.5 lb. x 24'0"

SIDE ELEVATION
FIG. 586



ELEVATION FLUE END
FIG. 587



ELEVATION BRIDGE WALL END
FIG. 588

FIGS. 586 to 588.—Elevations 100-ton storage (molding) reverberatory furnace.

in. deep. Its general construction is similar to that of a softening or refining furnace. Being a storage or molding furnace, in which the temperature is kept low, the sides and ends are not water-jacketed. The steel pan holding the brick-

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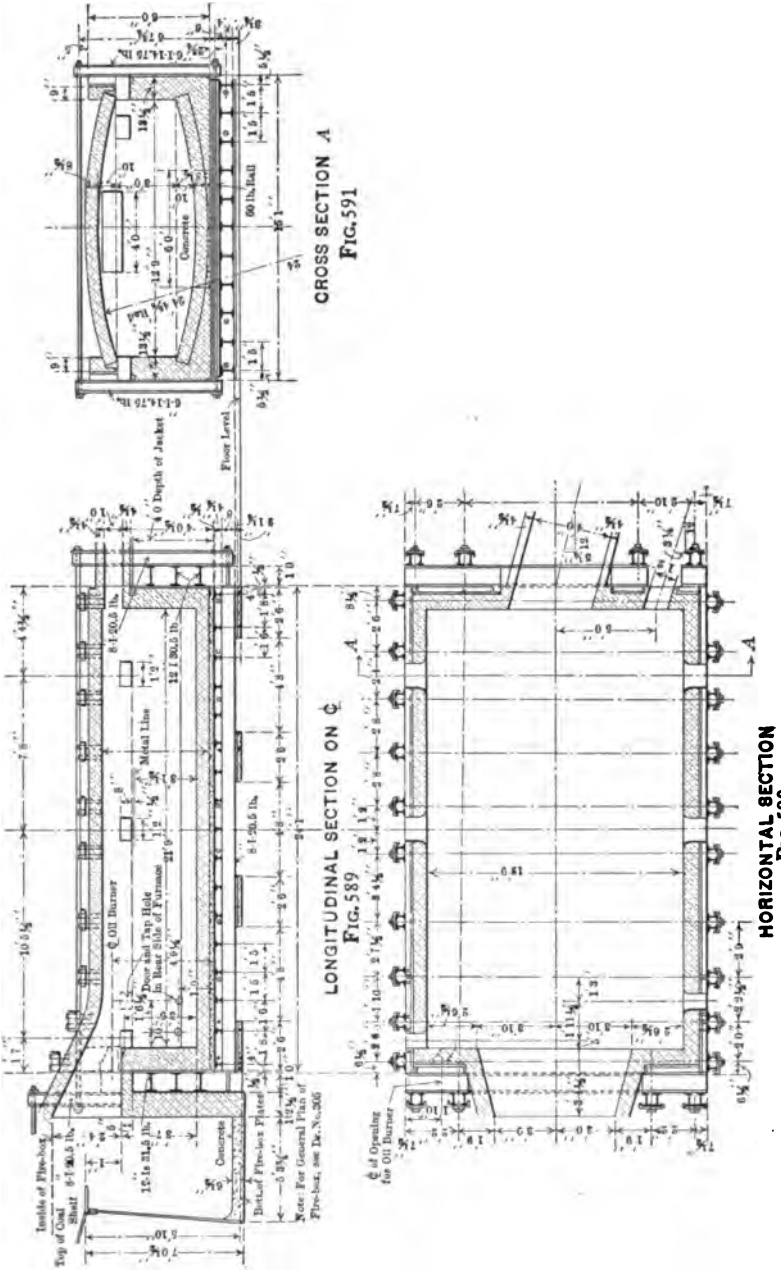
work stands free on cross-rails carried by longitudinal I-beams, and is heavily ironed. There are three doors to a side, and one end door. Provision is made



FIGS. 589 to 591.—Sections 100-ton storage (molding) reverberatory furnace.

for using an auxiliary oil-burner. The inverted arch forming the bottom rests on concrete for a distance of 3 ft. on either side of the center line; the remaining space is filled with red brick. The walls of the furnace are of fire-brick.

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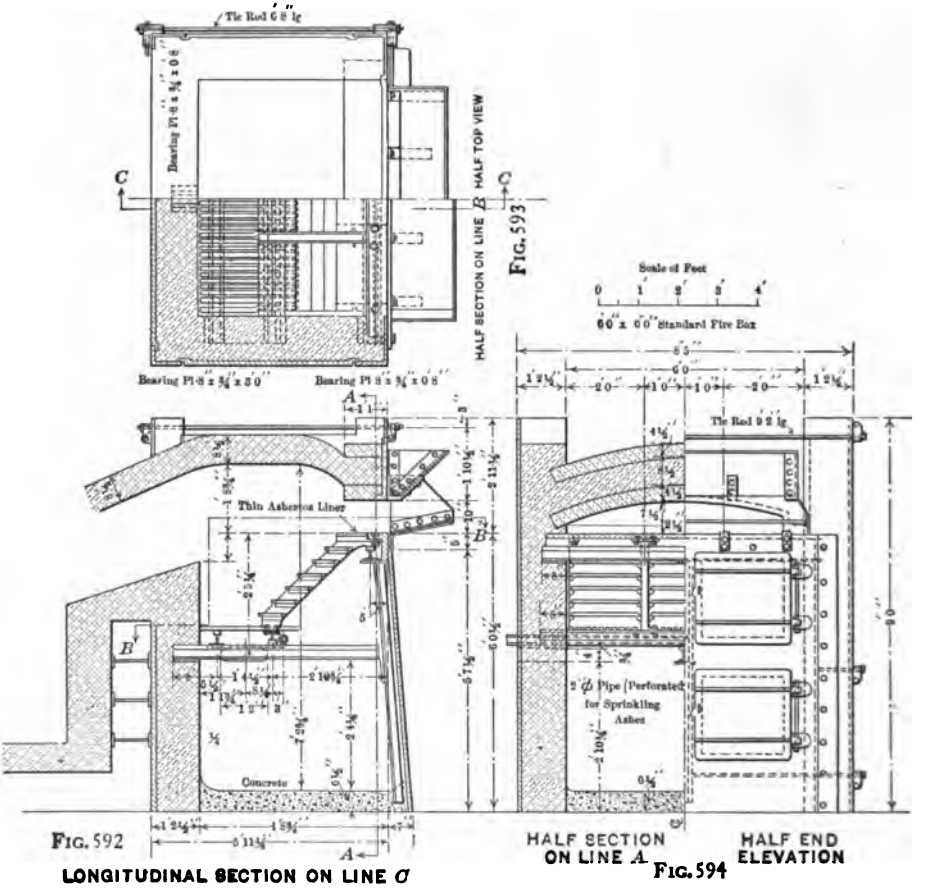
The fireplace, Figs. 592–594, has a step-grate, as the bituminous coal used is lignitic in its character, that is, it gives a long flame and has no caking power whatever. The fireplace is a 6- by 6-ft. standard which serves also the reverberatory furnaces for smelting drosses and skimmings (Figs. 627–630), and litharge (Figs. 631–634).


Tie Rod 6 8" lg
Bearing Pl 8 x ¾" x 2 0"
C
HALF TOP VIEW
HALF SECTION ON LINE B
FIG. 593
Scale of Feet
0 1 2 3 4
6'0" x 6'0" Standard Fire Box
Bearing Pl 8 x ¾" x 2'0"
Bearing Pl 8 x ¾" x 2'0"
A
Tie Rod 9 2 lg
Thin Asbestos Liner
B
2 ϕ Pipe (Perforated for Sprinkling Ashes
Concrete
FIG. 592
LONGITUDINAL SECTION ON LINE C
HALF SECTION ON LINE A
FIG. 594
HALF END ELEVATION

FIGS. 592 to 594.—Fire-place 100-ton storage (molding) reverberatory furnace.

The method followed in molding is an adaptation of the one used with copper refining furnaces in connection with the Walker casting machine.[1] It was in operation in 1913 at the National plant of the American Smelting and Refining Co., South Chicago, Ill.,[2] and has since been extended to other refineries. The lead is run from the furnace through a pipe, provided with regulating cock, into a cast-iron trough which swings on trunnions. A trough is a cylindrical vessel, 22 in. in diameter and 44 in. long, open at the top, which has five 1-in. pipes serving as spouts. When tilted forward by means of an air-lift, the lead

[1] Hofman, "Copper," 1918, 398.

[2] Pulsifer, *Min. Eng. World*, 1913, XXXIX, 154.

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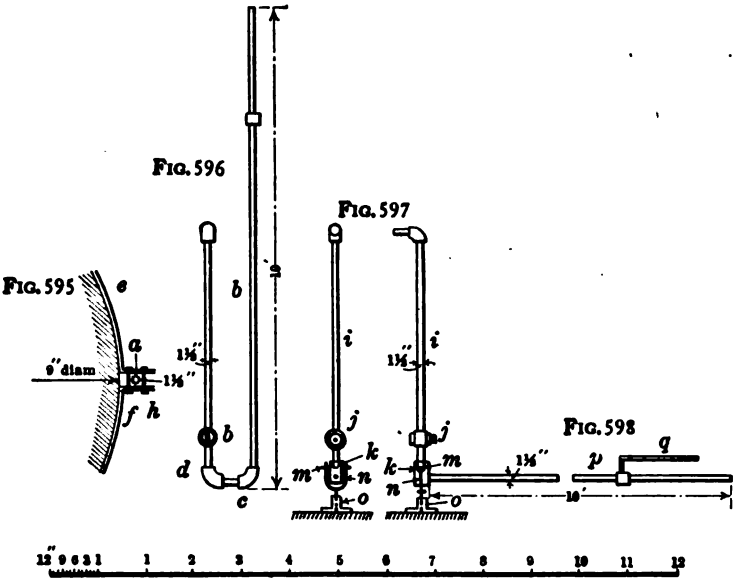
flows through the spouts into the molds. The molds, carried by an endless link-chain conveyor, travel in a horizontal direction in front of the trough, are filled, and at the same time sprayed with cooling water from below; the bars are skimmed two at a time by means of a three-pronged tool. The conveyor travels up an incline beyond the trough, whereupon the bars are trimmed and given the running lot-number by an automatic spring-hammer die. Arriving at the crest, the bars are discharged one after another on to a truck, which holds about 30 of them. In order to permit replacing a filled truck by one that is empty, there is left open in the conveyor a space which would be taken up ordinarily by five molds. This gives the time necessary for exchange of trucks.

In molding in this way, five or six bars at a time, either the conveyor has to stop in front of the trough while this is pouring, or the trough has to travel with



FIGS. 595 to 598.—Lead-molding apparatus for merchant kettle.

the conveyor while pouring, and to return to its original position at twice the normal speed. This latter mode of operating is found at the Omaha plant of the American Smelting and Refining Co.[1] The first plan is not so satisfactory for mechanical reasons as is the second; further the surfaces of the bars of lead are likely to be rough and broken when the conveyor has to stop before the lead has completely solidified. The rate of molding is 35 tons per hr., and the labor account is exceedingly small.

Molding from a *Storage* or *Merchant Kettle* was the common practice until the units became too large; it is in operation at most of the older refineries. The kettle has the same form and is heated in the same way as the one used for

[1] Pulsifer, *Min. Eng. World*, 1913, XXXIX, 457.

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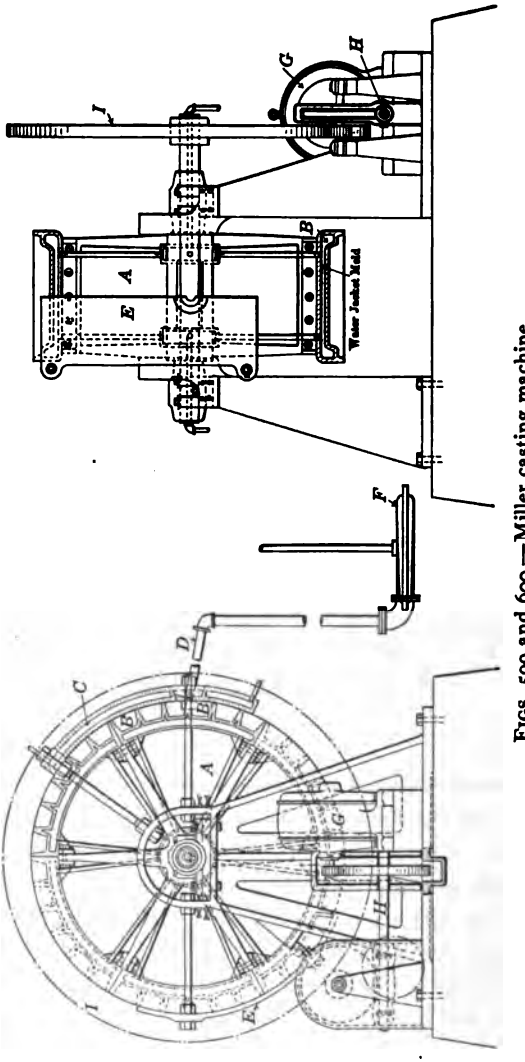
desilverizing. Until recently the Steitz siphon, Fig. 562, was used almost exclusively.

Two such molding apparatus are shown in Figs. 595–598. In Figs. 595 and 596, *a* represents the longer arm of the siphon, made of 1½-in. pipe, with



FIGS. 599 and 600.—Miller casting machine.

the cast-iron stop-cock, *b*. At the lower end it is joined by two elbows, with a nipple intervening, to the swinging-pipe, *b*. This can be turned down around the center, *c*, and when in that position it can be moved in a horizontal circle having its center at *d*. The siphon is filled just like the one shown in Fig. 562, with the

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exception that the handling is done with the swinging-pipe, *b*, instead of with a pair of tongs. As in molding, the swinging-pipe has to describe nearly a half- or three-quarters circle; the ordinary way of keeping the siphon in place by weighting with a couple of bars of lead is not sufficient. For this purpose two iron hoops, about 2 ft. 3 in. apart, pass around the brickwork of the kettle. The two ends of a hoop are bent, as shown in Fig. 595, and tied by a bolt. The vertical arm of the siphon, *a*, is held in place between the two hoop-ends, the fixed bolt, *f*, and the movable one, *h*.

Another arrangement for molding is shown in Figs. 597 and 598: *i* represents the longer arm of the siphon with the stop-cock, *j*; it discharges the lead into a 3-in. pipe, *k*, closed at the bottom. This has two trunnions, *m*, which swing in the bearings, *n*. They are joined to a pivot rotating in the socket, *o*. The pipe, *k*, is connected with the swinging arm, *p*, which is moved with the handle, *q*, over the molds. At some works, the pipe *k*, has been replaced by a trough. At others the bottom of pipe, *k*, is closed with a cap having a socket, into which the pivot, fastened to the floor, fits loosely. Thus a number of slight variations in detail are found at different works.

The latest device for molding lead from a kettle is the *Miller Casting Machine*[1] shown in Figs. 599 and 600. The leading parts are, a vertical cast-iron mold-wheel, *A*, with water-jacketed molds, *B*; a stationary curved water-jacket, *C*, on the feed-side, which extends above the center of the wheel and fits snugly the periphery of the revolving mold-wheel; a feed-pipe, *D*, in the stationary jacket, to deliver the lead into the molds as they pass underneath; and a shield, *E*, on the delivery side, to break the drop of the discharged bars as they fall on to a car or conveyor. Each mold has a recess in the lip (not shown) for the expulsion of the air into the next mold, and for receiving after the escape of the air a small amount of lead to serve as sample, which is discharged automatically when the bar drops from the mold. The lead is delivered to the feed-pipe from a kettle by means of a Miller pump, *F*, the excess flowing back into the kettle through a return-pipe, not shown. The mold-wheel is driven by a motor, *G*, making 1200 r.p.m., which is geared (6:1) to the shaft, *H*, of a worm, the wheel of which makes 4 r.p.m., and meshing with the drive-wheel, *I*, causes this to make 0.277 r.p.m. The machine casts 12.4 tons lead per hr. with three men; with a pump delivering from a central kettle through a radial trough seven men are required to do the work. The machine makes little dross, as the delivery and return lead are not exposed to the air; the bars are free from drosses, rough edges, and shrink-holes, and have a uniform weight.

254. Labor, Fuel, and Yield of Lead.—With small plants having 30-ton kettles, it has been customary to give in contract the operations of softening, desilverizing, and refining. With large units of 60- and 100-ton kettles this has become impracticable. Therefore each operation has its separate men who have to carry through their tasks in a given time. Thus, softening is placed on a 24-hr. basis, desilverizing and refining each on an 8-hr. basis. Molding and loading of lead is usually done by a separate crew in contract.

[1] U. S. Patient No. 1157794, Oct. 26, 1915; *Eng. Min. J.*, 1916, CII, 662.

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The fuel consumed in softening, desilverizing, and refining used to be 130 lb. per ton of lead bullion; this figure has been reduced to 50 lb.

The amount of lead recovered in the form of market lead varies with the purity of the lead bullion. It used to be 80 per cent. of the bullion charged, when this had not been carefully drossed before it reached the refinery. With dross-free bullion and the improvements made with the process, the yield in market lead at present is usually 90 per cent. of the lead bullion.

255. Treatment of Zinc Crust, General.[1]—Many methods have been tried and used for the treatment of zinc crusts. The only one which has survived is distillation, which originated with Parkes.

Smelting in the blast furnace (Flach process) with slag high in Fe, some matte and flux, and a low-pressure blast furnishes quickly a rich lead bullion, but sacrifices the zinc, and is accompanied by large losses in silver and lead.

Cupellation of the crust is more crude than smelting in the blast furnace.

Fusion with alkali chlorides with the idea of converting the zinc into soluble chloride was a failure.[2]

Treatment of partly liquated crust with steam in a kettle at an elevated temperature, with the aim of oxidizing the zinc (and incidentally some lead) and thus obtaining a rich bullion and a mixture of lead and zinc oxides, has been in operation in several European plants. The rich lead used to be cupelled, and the oxide mixture added gradually to the charge. Later Schnabel leached the oxides with ammonium carbonate, precipitated any Cu present with Zn, filtered the solution, recovered the zinc from it by boiling with steam as a basic carbonate which, after calcination, gives a zinc white with a shade of yellow, and cupelled the rich lead, adding the lead residue, freed from Zn, to the charge. The process[3] was in operation at several European plants, but has become obsolete.

256. Distillation of Zinc Crust, General.[4]—This process, first used by Parkes, has undergone many improvements and has become the one universally used since Balbach[5] first used graphite retorts. The method, therefore, often bears his name. Retorting in vacuo[6] has been suggested, which would permit carrying out the process at a much lower temperature than is the case under atmospheric pressure; but so far it is merely a suggestion.

A furnace, to be suited for the process, must permit the raising and sustaining of a high temperature, and at the same time be of such a construction that a

[1] Kerl, B., "Grundriss der Metallhüttenkunde," Felix, Leipsic, 1881, 314.
Roswag, "Désargentation de plomb," Dunod, Paris, 1884, 296.
Schnabel, C.-Louis, H., "Handbook of Metallurgy," Macmillan, London, 1905, I, 664.

[2] Zeiller-Henry, *Ann. Min.*, 1870, XVII, 447; *Berg. Hüttenm. Z.*, 1871, XXX, 422.

[3] Details: Schnabel-Louis, *op. cit.*, 679.

[4] Works cited in §244.
Eilers, *Tr. A. I. M. E.*, 1874–75, III, 314.
Roesing, *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1886, XXXIV, 91; *Berg. Hüttenm. Z.*, 1886, XLV, 421.

[5] Eurich, *Tr. A. I. M. E.*, 1912, XLIV, 741.

[6] Turner, *J. Inst. Met.*, 1912, VII, 105.

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broken retort can be readily exchanged, and the rich bullion that has run out, quickly and completely collected. Of the different forms, three may be cited.

Furnaces have been fired with coal, coke, oil, and producer gas. Coal-firing has become about obsolete, as it is laborious and requires considerable skill to keep the temperature uniform and even; coke, oil, and producer gas work satisfactorily. At present, oil is the leading fuel in the United States. Producer gas found favor in Germany[1] and Australia.[2]

The retorts used have been given different forms, *e.g.*, that of a crucible at Freiberg, and of a cylinder at Tarnowitz;[3] usually they are pear shaped. The condensers are either of clay or iron; partly corroded retorts are frequently used as condensers.

The older forms of furnaces and retorts are given by Schnabel-Louis,[4] Collins,[5] and others. In the United States the Faber du Faur furnace has outlived all other forms, especially after the patents of the inventor had expired.

257. Faber du Faur Retorting Furnace.—Two forms, the coke and the oil fired, may serve as examples. Formerly practically all furnaces were coke fired. The many advantages that oil-firing offers, such as uniformity of heat, absence of clinkers, facility of handling, etc., have pushed coke-firing into the background to such an extent that at present it forms the exception.

1. *The Coke-fired Furnace* (Figs. 601–605).—This is a crucible furnace of cubical form, built into cast-iron frame-work that swings on trunnions, in order that the furnace may be turned over and the contents of the retort emptied. The furnace is closed at the top by an arched roof, which usually has one opening, the charging opening, for the coke. The products of combustion pass off through a flue, which is generally placed at the back, as in the drawing, but sometimes at the side, and occasionally in the roof. At the front is an opening for the neck of the retort. The bottom is formed by two sets of rectangular wrought-iron grate bars placed on edge. The retort rests on a small brick pillar, which is supported either by a brick arch (as in the figure) or by an iron plate running from front to back, and protected from the heat by two courses of fire-brick. The rotation of the furnace is effected by a worm-gear, or sometimes simply by means of a lever. The furnace in its original form of 1874, as represented by the drawing, was about 4½ ft. cube, and was intended for a pear-shaped retort holding 250 lb. of liquated crust. It was lined with a full course of brick, except at the front, where the brickwork was 4½ in. thick.

At present the retorts, though retaining their original form, are made larger and thinner; they hold from 1000 to 1500 lb. of crust; the furnace has retained its original size, but is lined on all sides with a half course of brick. The old retorts (Fig. 601) were made of raw and burned clay mixed with about 25 per cent. of graphite to protect the clay from the corrosive action of the lead, and

[1] Roesing, *loc. cit.*
[2] Bayly, *Tr. Austral. Inst. Min. Eng.*, 1907, XII, 91, 92.
[3] Roesing, *loc. cit.*
[4] *Loc. cit.*, 665.
[5] "Metallurgy of Lead," 416.

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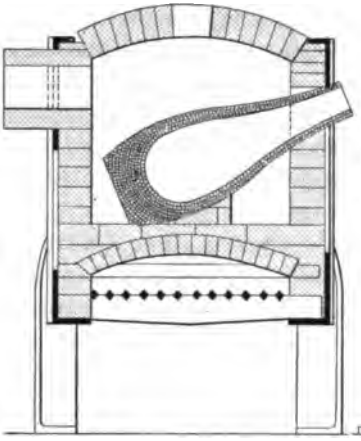

FIG. 601

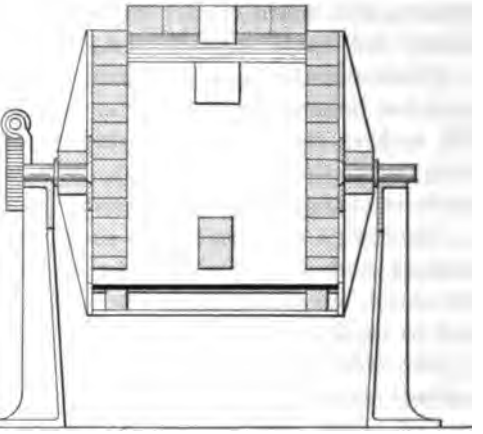

FIG. 602

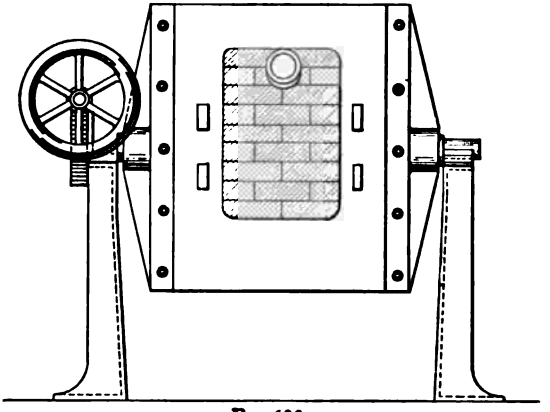

FIG. 603

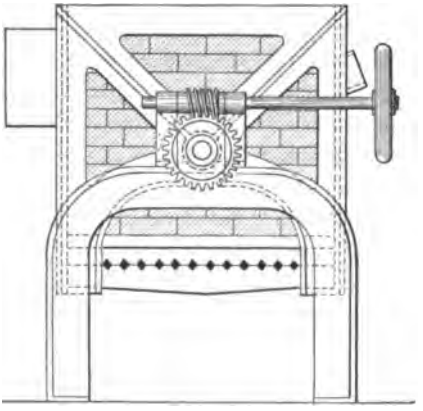

FIG. 604

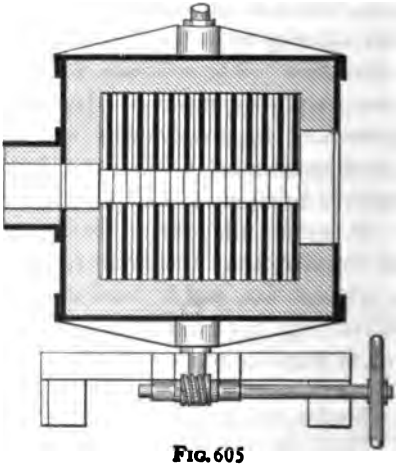

FIG. 605

FIGS. 601 to 605.—Faber du Faur retorting furnace, coke-fired.

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were very thick at the bottom; at present they are made of graphite to which sufficient clay (45 per cent.) has been added to give strength and stability to the retort. A 1000-lb. retort is 36 in. high, 8 in. wide at the neck, 18 in. at the belly, and 13 in. at the bottom. It is 1⅛ in. thick at the neck, and increases to 2 in. at the bottom. A so-called 1500-lb. retort is shown in Fig. 606. It is advisable, however, to charge not more than 1300 lb.

A set of retorting furnaces is arranged in two ways. They are either placed on both sides of a horizontal flue in such a way that the openings into it shall not be opposite each other, or they are built, say eight, around a central stack, each flue extending into the stack and then continuing upward for a few feet. With both arrangements obstructions of the draft are avoided.

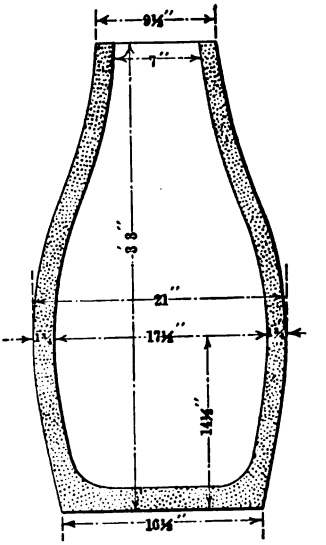

FIG. 606.—Graphite retort, capacity 1500 lb. charge.

2. *The Oil-fired Furnace.*[1]—A modern furnace with full details is shown in Figs. 607–612. Again there is seen a cast-iron frame work swinging from trunnions, but lined with 9 in. fire-brick at the sides and 4½ in. at front and back. It is fired with oil from a Billow Long-flame Burner, Class F R[2], represented in elevation and section in Fig. 613. This is a brass, chamber oil-burner,[3] 12 in. long, with central oil compartment provided with a needle-valve and enclosed by the air division. The chamber has fixed vanes to effect an intimate mixture of atomized oil and air; the outlet nozzle is ⅝ in. in diameter. Oil is held at a pressure of 10 lb. per sq. in., and air at one lb.

The retort is 3 ft. 2⅜ in. long and 22⅞ in. in diameter at the belly; the walls are 3 in. thick at the bottom and taper to 2 in. at the neck. The vessel holds 1350 lb. crust mixed with a little charcoal; it rests on a fire-brick block. It will be noticed, Figs. 609 and 610, that the retort is placed ⅞ in. beyond the center line of the furnace in order that the heating space near the burner may be larger than the one opposite, and thereby the distribution of heat made even. The correct placing of the retort is of greatest importance as regards life of vessel and result of distillation. Fig. 607 shows in dotted lines the cast-iron condenser held in place by a chain suspended from two hooks (Figs. 607, 608, 612). At the front of the furnace, Figs. 607 and 608, next to the floor, are two square openings for tapping any lead that may have escaped from a leaking retort.

258. Condensers.—The condensers used for collecting the zinc differ very much in form and material. Some are simply old retorts; others are plumbago

[1] A drawing of the oil-fired furnace at Maurer, N. J., has been published in *Eng. Min. J.*, 1907, LXXXIII, 84.

[2] National Supply Co., 416 West Grand Ave., Chicago Ill.

[3] Hofman, "General Metallurgy," 1918, 336.

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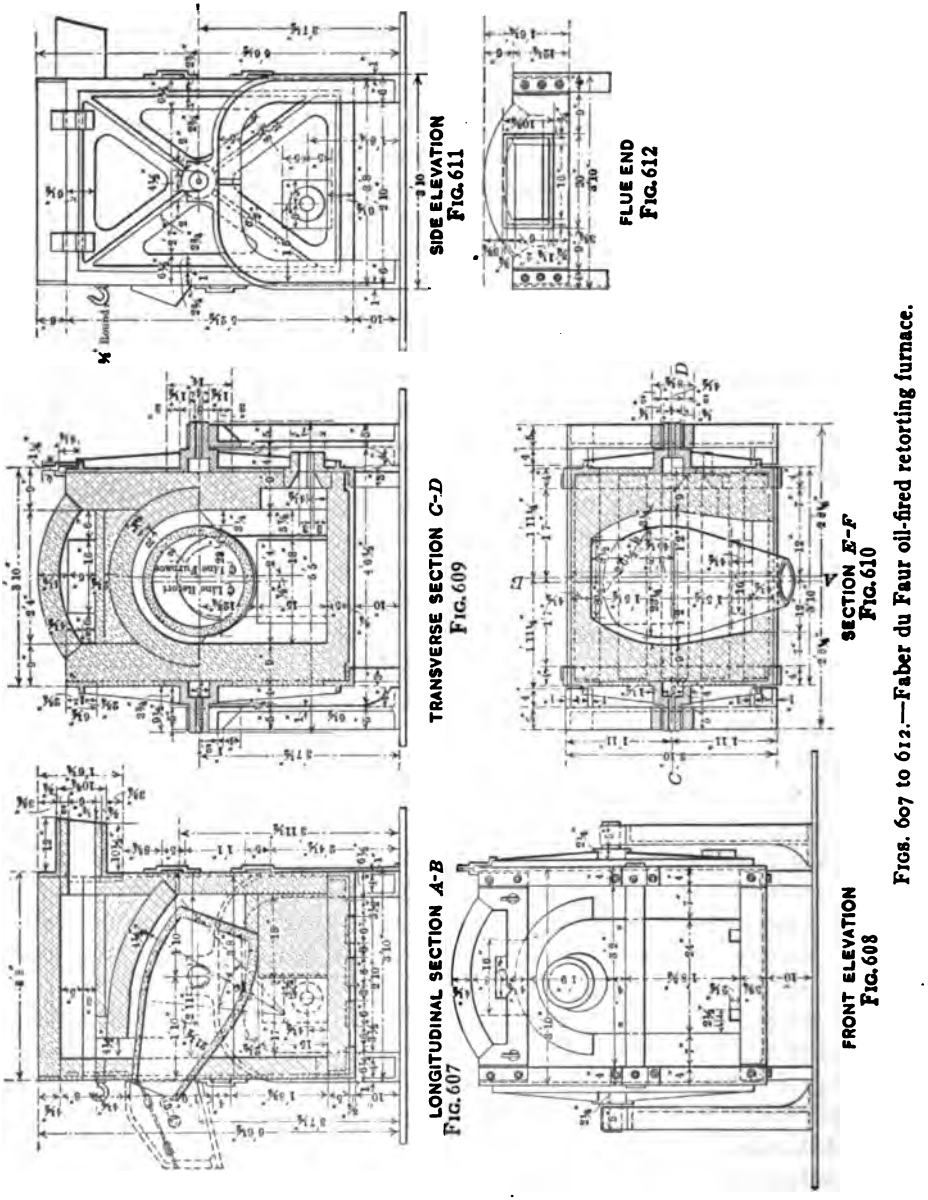
crucibles (diameter at bottom, 7 in.; at top, 11 in.; height at front, 18 in.; at back, 22 in.; thickness, 1 in.). Again, they are made of cast iron having the form of a truncated cone. One condenser of this class is 2 ft. long, and has


SIDE ELEVATION FIG. 611
FLUE END FIG. 612
TRANSVERSE SECTION C-D FIG. 609
SECTION E-F FIG. 610
LONGITUDINAL SECTION A-B FIG. 607
FRONT ELEVATION FIG. 608

FIGS. 607 to 612.—Faber du Faur oil-fired retorting furnace.

handles on either side by which it is suspended on two hooks from the iron frame of the furnace. A conical condenser is also made of clay; it is about 3 ft. long and supported by a tripod. Another form is that of a sheet-iron cylinder

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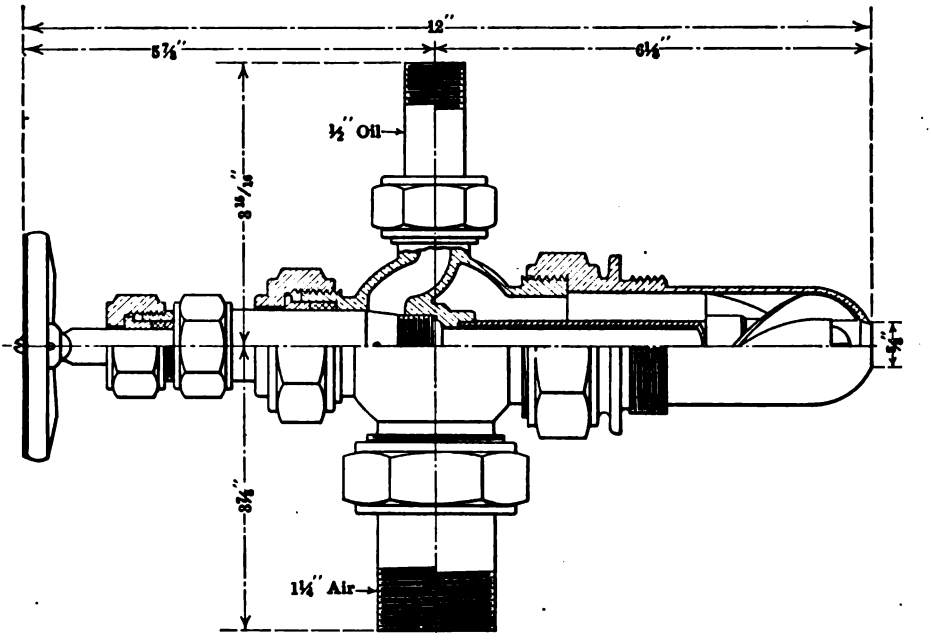
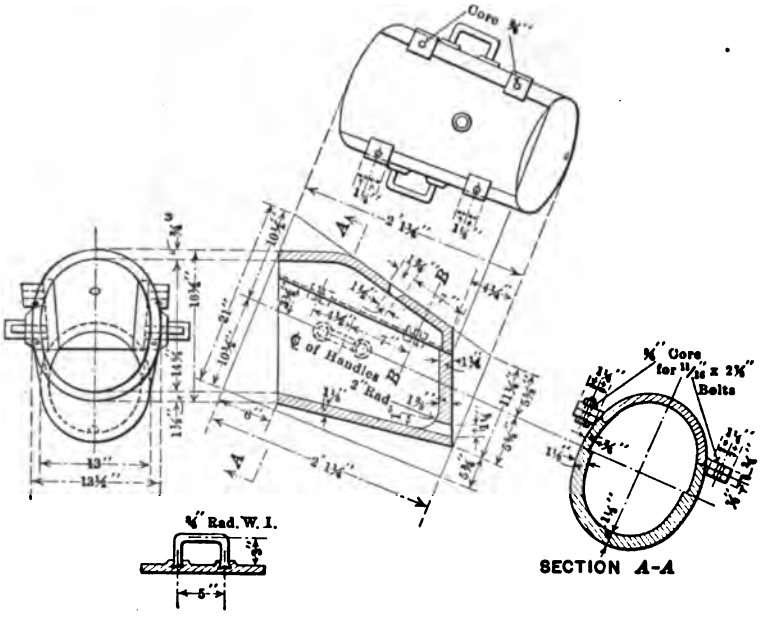


FIG. 613.—Billow long-flame burner.



FIGS. 614 to 618.—Two-piece cast-iron condenser.

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lined with specially molded fire-bricks. At the base it has, in addition to the tapping-hole, two small pivots to hold a thin chain which is hooked to the frame of the furnace, thus supporting the condenser. It is claimed by some refiners that an iron condenser furnishes a larger yield in zinc than one made of clay.

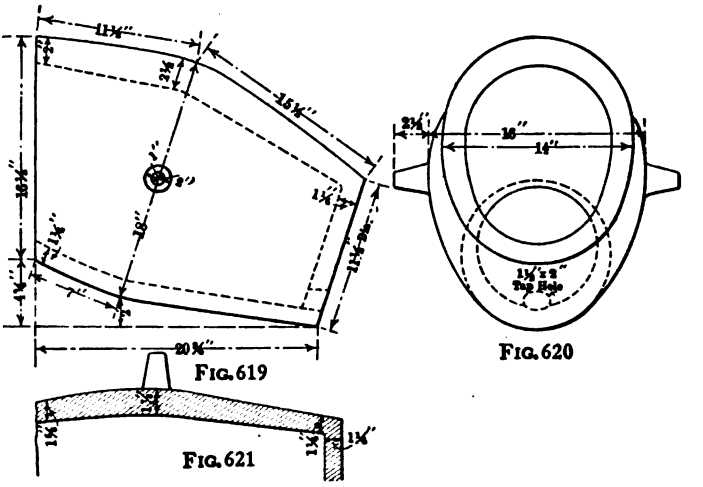
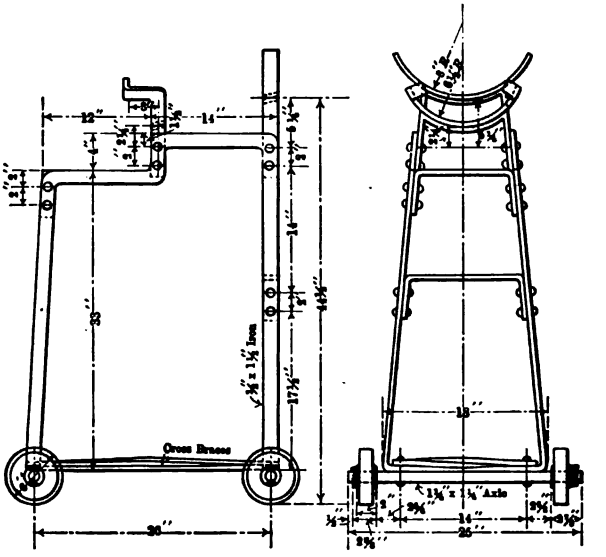


FIG. 619 FIG. 620

FIG. 621

FIGS. 619 to 621.—Single-piece cast-iron condenser.



FIGS. 622 and 623.—Truck and stand for condenser.

In Figs. 614–618 are given the details of the two-piece cast-iron condenser indicated by dotted lines in Fig. 607. It has a peculiar form which is the result of experimentation. It is made up of two pieces bolted together, the lower being heavier than the upper; there are handles on the sides, a 1¼-in.

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tap-hole, and a 1½-in. port for the passage of gases and fumes. At one plant the lower part is of cast steel. The manner of support has been given already.

Figs. 619–621 show another condenser of form similar to the one given in Figs. 614–618, but of more simple construction. It is a single-piece casting with handles on the sides, the usual tap-hole, but no port for the escape of gases. The condenser is carried by a truck-stand of flat iron given in Figs. 622 and 623. The front view, Fig. 623, shows the curved support for the condenser; the side-view, Fig. 622, presents a horizontal tray which carries the mold into which is tapped the condensed zinc.

The condensed zinc is rarely allowed to run off continuously. Usually there is a tap-hole closed by a clay plug or a conical piece of coke, and the condensed zinc is discharged into a suitable mold in 3 or 4 hr. after the distillation begins. It has been found that more zinc will collect in the condenser if this contains some liquid zinc. Most condensers have a second opening on the upper side for fumes to pass through while the distillation is going on. They go into a sheet-iron pipe leading to the main flue. It has always been considered essential to have this second opening if the distillation is to proceed in a satisfactory way; but at some works using an old retort as condenser the second opening has been dispensed with, the condenser being put in place only when the distillation begins, and not, as is customary, when the retort is well filled.

259. The Method of Working.—The method of working is about the same whichever furnace is used. When a new retort has been put in place, it is carefully warmed by burning in it for 24 hr. slabs of wood, and brought to a dull red heat. It is then ready for the oil-burner and for charging. The zinc crust, mixed with from 1 to 3 per cent. of charcoal, is brought in an iron wheelbarrow or car to the front of the retort and charged with a trough-shaped scoop. The temperature is then quickly raised to about 1200° C.; the crust softens and sinks in the retort after half an hour, is pushed down, more crust is added, and this is repeated till the retort is well filled. The upper rim of the condenser receives a heavy lute of clay, is passed over the neck of the retort, and is made to adhere to the front wall of the furnace; it is suspended or rests on its support. About an hour after charging, blue powder and then metallic zinc begin to collect in the condenser.

At atmospheric pressure zinc boils at 918° C.[1] In laboratory experiments for the recovery of zinc from cyanide precipitate, Clevenger[2] found that at 1200° C. there was expelled 90 per cent. and at 1296° C. as much as 99.6 per cent. of the zinc present; also that it was important to raise the temperature quickly and to keep it constant when it had attained its proper height.

The zinc vapor carries over with it some lead if the temperature is too high. Juretzka,[3] in distilling impure zinc, found that at 1000° C. some lead was carried over, and that at 1100° C. the condensed zinc vapor contained 0.28 per cent. Pb.

[1] Berthelot, *Compt. rend.*, 1902, CXXXIV, 705.

[2] *Tr. A. I. M. E.*, 1904, XXXIV, 891; *Eng. Min. J.*, 1914, XCVIII, 109.

[3] *Chem. Z.*, 1916, XL, 885, 894.

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In distilling zinc-silver alloy, Rose[1] experienced losses in Ag averaging 8.46 per cent.; Friedrich[2] found losses to range from 0.4 to 5.8 per cent. Ag, the losses increasing with the quantity of zinc and the temperature. In working zinc-silver-lead crusts, the large amount of Pb and the small percentage of Zn change conditions. Some Ag passes over with the Zn; the collected zinc assays from 4 to 12 oz. Ag per ton depending upon the rate of distillation. With 8000-oz. retort bullion the collected zinc has shown assays as high as 24 oz. Ag per ton; ordinarily a 1200-lb. charge gives zinc with from 4 to 5 oz. Ag per ton.

The work of Rose[3] and Friedrich[4] with zinc-gold alloys shows that the loss experienced is chiefly mechanical.

The behavior of tellurium,[5] which readily enters the zinc crust in desilverizing, is of interest, as long as telluride gold ores are treated by lead smelters. Table 141 gives some analytical data of the products obtained in retorting tellurium-bearing zinc crusts from Pertulosa, Italy. The three successive zinc taps, with their increasing percentage of Te, show that Te is not as easily

TABLE 141.—DISTILLATION OF ZINC CRUST CONTAINING TELLURIUM

Products	Te(*a*)	Se	As	Sb	Sn	Bi	Cu	Cd	Fe	Zn
First zinc tap..	0.0030	.....	.....	.....	.....	.....	.....	.....	.....	.....
Second zinc tap	0.0050	.....	.....	.....	.....	.....	.....	.....	.....	.....
Third zinc tap..	0.0150	.....	.....	.....	.....	.....	.....	.....	.....	.....
Skin of zinc, lining condenser	0.0370	.....	Trace	Trace	.....	.....	.....	.....	0.3270	.....
Zinc telluride...	30.0000	.....	0.0180	0.0220	.....	.....	.....	.....	0.4090	(*a*)67.0240
Blue powder...	3.4600	.....	.....	.....	.....	.....	.....	.....	.....	.....
Retort bullion..	0.5156	Trace	1.4095	0.3019	None	0.0237	2.2477	0.0052	0.2205	....0.8537
Gas from condenser	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

Products	Ni, Co	S	Ag	Au	Pb	CO_2 (*b*)	CO (*b*)	N (*b*)	C
First zinc tap.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Second zinc tap.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Third zinc tap.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Skin of zinc, lining condenser	.....	Trace	0.0060	.....	1.4700	.....	.....	.....	Trace
Zinc telluride.....	Trace	.....	0.0040	.....	2.5230	.....	.....	.....	Trace
Blue powder.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Retort bullion.....	0.0068	Trace	5.4616	0.0564	(*a*)88.8974	.....	.....	.....	.....
Gas from condenser.....	.....	.....	.....	.....	.....	1.4	55.2	43.4	.....

(*a*) By difference. (*b*) Volumes.

[1] *Tr. Inst. Min. Met.*, 1904–05, XIV, 394.
[2] *Metallurgie*, 1908, V, 593.
[3] *Loc. cit.*, 392.
[4] *Zt. angew. Chem.*, 1903, 269.
[5] Heberlein, *Berg. Hüttenm. Z.*, 1895, LIV, 41; *Min Ind.*, 1895, IV, 480.

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distilled as is the Zn. The fact that the zinc adhering to the iron condenser as well as the retort bullion, retains considerable Te, substantiates this. Samples of the carmine colored telluride of zinc adhering to the iron condenser, and to some of the zinc, showed as much as 30 per cent. Te, the rest being made up by Zn and the impurities shown in the table. Telluride of zinc changes its color to a grayish black in a few days if exposed to daylight. If it comes in contact with air while hot it is quickly oxidized; the reducing atmosphere in the condenser permits the condensation of the alloy. Some alloy, however, is oxidized, as seen from a selected sample of blue powder, which is very rich in tellurium.

The distillation is finished after from 6 to 8 hr., according to the size of the charge, the percentage of zinc in the crust, and the draft of the furnace. Too much emphasis cannot be laid upon the draft; especially with coke-fired furnaces. The slightest obstruction means a failure in distilling off the zinc in the required time, and many consequent disturbances, inconveniences, and losses. The main points to be looked after during the operation are the quick raising of the temperature, and the keeping it high. The higher the temperature, the more rapid will be the distillation, the better the output of metallic zinc, and the lower the percentage of zinc remaining in the rich lead; but forcing a charge is likely to decrease the yield of metallic zinc, and increase the amounts of lead and silver which pass over with the zinc. If the temperature is lowered, blue powder forms, and some of the crust floating on the lead may harden and be suddenly broken by the zinc vapors developed beneath the crust, when the heat is raised again, and thus cause an explosion. The effect of this is to loosen the condenser from the neck of the retort. At most works it is the practice to introduce an iron rod, at certain intervals, through the upper opening in the condenser, to free the neck of the retort with it from oxidized zinc, and then stir up the crust floating on the lead. But the saying that the condenser ought to smoke well if the distillation is to go on satisfactorily is wrong because a condenser with only one opening, excepting the tap-hole, which is kept closed, gives excellent results.

The zinc is tapped at intervals from the condenser; when it ceases to collect in the condenser the distillation is finished. The last zinc is then removed, and the condenser taken off and scraped clean. In the meantime the vapors in the retort pass off into the air. This is assisted by throwing a few chips of wood into the retort, which is now to be emptied either into an ordinary slag-pot, lined with brick to prevent the hot metal from cracking the cold bottom, or into a tilting pot. Such a pot is shown in Figs. 624–626. It is a cast-iron vessel, 28 in. in diameter, provided with trunnions which are carried by supports of a car 3 ft. long. The vessel, lined with 2½ in. of fire-brick, is tilted by means of a worm-gear. While the lead is running from the retort, the sample for assay is taken. The lead is then allowed to cool and, after the removal of the dross from the surface, is ladled or poured into small molds, so as to obtain bars of suitable size for the subsequent treatment in the English cupelling furnace. The dross is also cast into bars and added to the cupelling charges.

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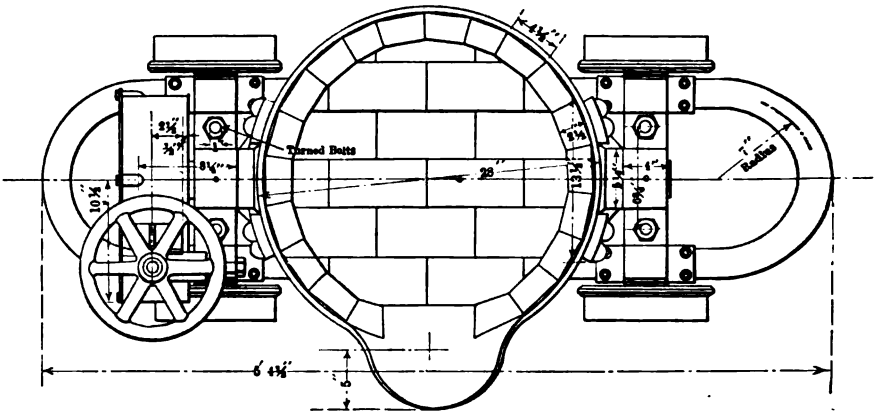

FIG. 624 PLAN

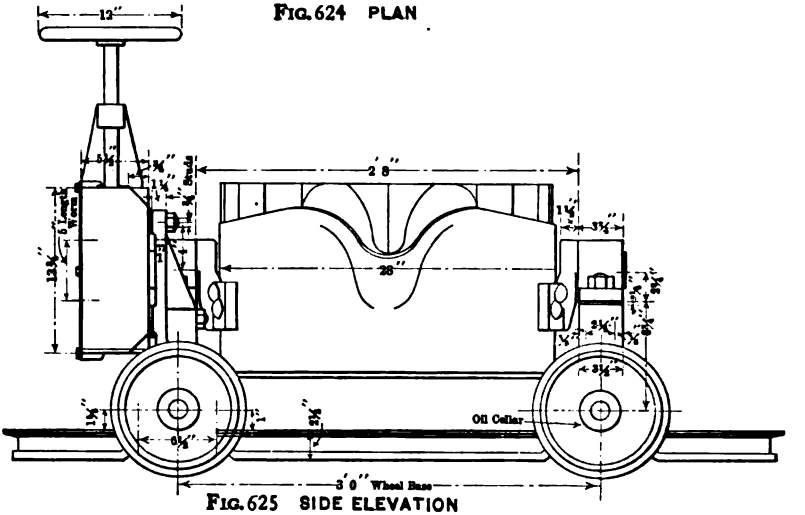

FIG. 625 SIDE ELEVATION

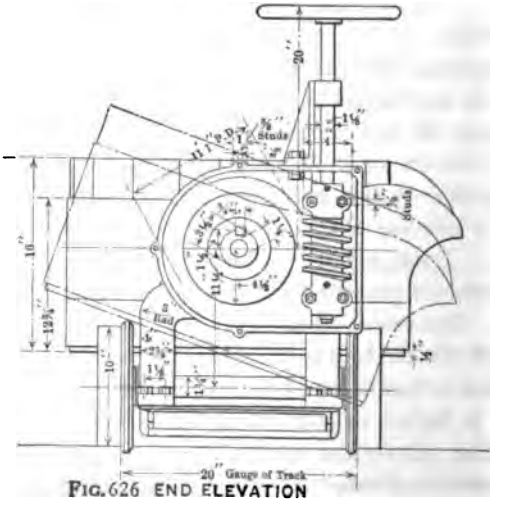

FIG. 626 END ELEVATION

FIGS. 624 to 626.—Tilting pot for retort bullion.

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At many works, the pot with the retort bullion is raised from its supports by means of an overhead crane, a traveling air-lift, or a differential pulley, transferred to the cupelling furnace, and its content poured into the furnace, thus doing away with drossing, molding, etc. This mode of operating was started in 1901 by F. B. F. Rhodes at the National Works in South Chicago, Ill.[1] The residue in the retort, consisting of slag and charcoal, is removed with an iron scraper. It is essential for the life of the retort that it be well cleaned after each distillation. The retort is now raised, and some fine charcoal thrown into it to prevent the oxidation of the globules of lead adhering to the sides; it is then turned back to its normal position, and is ready to be charged again. With a coke-fired furnace the grate bars are cleaned, clinkers adhering to the furnace walls or sides of the retort are removed, and fresh coke is added, making the furnace ready for the next charge. The time required for discharging and refilling is about 20 min. In scraping, care must be taken to do it gently, so as not to wear off the lower side of the retort. To avoid this, as well as to change the line of contact with the surface of the lead, the position of the retort is sometimes changed, after it has worked about 25 charges, by turning it 180°.

The tools required by one man in retorting are: two scrapers (6 ft. long, 5/8-in. round iron flattened out at one end to the width of 3 in. and bent up 3 in.) to stir the contents of the retort after distilling and to rake out the residue after tilting; two pokers (5 ft. long, of 7/8-in. iron) to remove the clinkers from the grate; one bar (8 ft. long, 3/4-in. steel) to break off clinkers from the walls; one scoop to feed the coke; one ladle (6 in. in diameter and 3 in. deep, with a 4-ft. handle); two slag-pots lined with brick; a wheelbarrow to receive the zinc crust; one scoop and shovel to charge the retort; two molds for each retort to receive the zinc; 10 bullion molds.

A method of doing away with the usual procedure of retorting has been suggested by Roesing.[2] It is to melt the crust in a basic-lined retort, to charge some fuel for the expulsion of air, and to pour in melted pig iron. This is to volatilize the zinc which is collected in a condenser; then the pig iron is poured off, to be used again, and followed by the retort bullion.

260. Results.—The weight of the charge is from 1000 to 1300 lb. of zinc crust, and it takes from 6 to 8 hr. for one operation; less time is required in winter than in summer. The crust yields from 70 to 80 per cent. of Pb containing from 4 to 10 per cent. of Ag and from 0.75 to 1.50 per cent. of Zn. The zinc recovered in the form of metal is about 10 per cent. of the weight of the crust, and that in the form of blue powder about 1 per cent. Of all the zinc required for desilverizing, over 60 per cent. used to be recovered to be used again in the kettles. This has been raised to 80 per cent. by the use of the Howard stirrer and press, and by oil-firing. The quantity of dross varies a great deal; from 5 to 8 per cent. of the weight of the crust was a fair figure, before the Howard improvements. The blue powder formed amounts to from 12 to 18 lb. per charge. The addition of 1 lb. salt and 4 lb. sand to a 1200-lb. charge is

[1] Eurich, *Tr. A. I. M. E.*, 1912, XLIV, 750.

[2] *Berg. Hüttenm. Z.*, 1890, XLIX, 369.

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said to reduce the amount from 18 to 12 lb. The blue powder is either returned to the retort, or sold to zinc works after screening.

TABLE 142.—DISTILLATION OF ZINC CRUST

Test	Concentration	Per cent. Zn recovered	Pounds Zn recovered	Material	Lb.	Per cent. Pb.	Per cent. Zn	Per cent. Doré	Per cent. Sb	Per cent. As	Per cent. Cu	Per cent. SiO_2	Per cent. Fe	Per cent. S
Zinc crust from pure silver-lead; no impurities.	14 to 1	66	399	Zinc crust...	2700	64.4	23.0	9.7	...	...	....	...	...	....
				Retort metal	1671	....	1.5	12.3	...	...	....	...	...	....
				Blue powder.	82	6.2	86.0	0.7	...	...	....	...	...	....
				Slag.....	548	54.4	22.9	10.2	...	...	....	1.9	1.8	.66
Zinc crust from lead bullion: 0.1 per cent. Sb.	22 to 1	79	589	Zinc crust...	2551	56.0	29.3	13.1	0.1	...	....	...	...	....
				Retort metal	1659	79.0	2.3	18.2	0.1	...	....	...	...	....
				Blue powder.	84	6.2	85.5	0.9	0.1	...	....	...	...	....
				Slag.....	194	50.9	22.9	14.1	Tr	...	....	1.7	1.7	0.46
Zinc crust from lead bullion; 0.3 per cent. Sb.	30 to 1	88	175	Zinc crust...	1015	70.3	19.8	8.2	0.2	...	....	...	...	....
				Retort metal	730	....	0.9	10.3	0.2	...	....	...	...	....
				Blue powder.	5	3.5	88.9	0.6	...	...	....	...	...	....
				Slag.....	99	61.5	13.6	7.5	Tr	.....	.....	1.0	2.0	0.32
Zinc crust from lead bullion; 0.1 per cent. As.	11 to 1	89	503	Zinc crust...	2580	67.7	21.5	7.6	...	0.5	....	...	...	....
				Retort metal	1803	....	1.3	9.6	...	0.6	....	...	...	....
				Blue powder.	7	6.2	77.6	0.4	...	Tr.	....	...	...	....
				Slag.....	244	60.0	12.5	9.4	....	0.8	....	1.7	...	....
Zinc crust from lead bullion; 0.2 per cent. Cu.	9 to 1	60	209	Zinc crust...	2344	80.2	12.0	7.0	...	...	1.00	...	...	....
				Retort metal	1403	90.0	0.7	7.6	...	....	1.24	...	...	....
				Blue powder.	14	10.0	75.6	0.2	....	...	Tr.	...	...	....
				Slag.....	686	63.1	18.4	5.8	....	....	0.65	0.5	...	....

In Table 142 are given results obtained in working crusts free from impurities, and crusts with varying amounts of Sb, As, and Cu. Column 1 gives the concentration of silver in lead effected by the Parkes process; it is seen to range from 9 to 30 per cent. and is usually near 30 per cent. The recovery of zinc is seen to cover the wide range of 60 to 89 per cent.; a common figure is 80 per cent. The high zinc-content of the slag (dross), 12.5 to 22.9 per cent., shows the direction in which lies a considerable loss of zinc. In working off the slag (dross) in the cupel, the high zinc-content of the charged material is very apparent. The slag (dross) also shows high values in doré. Newton[1] calls attention to the effect that Cu sometimes has in forming, with low distillation temperatures, lumps which assay as high as 11,000 oz. Ag (Au) per ton, when the retort bullion has a value of 3000 oz. Blakemore[2] emphasizes the effect Cu has on the presence of Au in dross.

The impurities Sb, As, and Cu seem to be evenly distributed in crust and retort metal, but appear only as traces in the blue powder.

A retort lasts now 40 charges; formerly 25 charges was considered a good average. For every ton of crust, 1100 lb. of coke or 40 to 60 and even 70 gal. oil, with air as atomizer, are required. To use a good quality of coke of uniform

[1] *Tr. A. I. M. E.*, 1915, LI, 791.

[2] *Tr. Austral. Instit. Min. Eng.*, 1898, V, 247.

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size (egg coke), although more expensive at first, is cheaper in the end, when the life of the retort and the better results obtained are considered. One man attends to from two to four furnaces in an 8-hr. shift.

As there is likely to be some leakage in a retort, which escapes attention, it is advisable not to throw away the coke ashes, but to add them to the blast furnace charge. Flue-dust from retort ovens gave Iles Zn 12.2 per cent., Ag 16.8, and Au 0.20 oz. per ton. As stated, reduced oil has replaced coke or coal in most cases. With a 1000-lb. charge, 550 lb. coke in 6 hr. were replaced by 52 gal. oil in 6 hr. with steam injection, and 26 gal. in 5 hr. with air injection; in both cases the Reid burner[1] was used.

261. Treatment of Intermediary Products, General.—The intermediary products in a refinery are formed in working the lead bullion, when it carries precious metal (softening-furnace dross and skimming, kettle dross, rich lead, metallic zinc, retort dross and blue powder, litharge, etc.), or they arise in the purification of desilverized lead (refining skimmings and polings) and in the casting of refined lead, which is free from silver. It is important that they be converted into marketable products as quickly as possible. Their treatment has to be carried on simultaneously with the main operations in order that they may not accumulate, and thereby increase the outlay of capital necessary to run the plant, thus eating up interest. Some accumulation cannot be prevented, as it is necessary in smelting to have on hand certain amounts of material to permit running furnaces for given periods of time.

262. Softening-furnace Dross and Skimming.—The lead bullion charged into the softening furnace at present has been freed from dross at the smeltery so that not enough dross is taken off in the softening furnace to make it an object to treat it by itself. It is therefore not kept separate from the skimming. Undrossed bullion is usually melted in a kettle, and the dross freed from adhering lead by means of a Howard press.

Formerly drossy bullion was charged into the softening furnace, and the dross, after removal by skimming, freed from entangled lead in a separate reverberatory furnace.

In liquating, the furnace is charged and heated until the lead that has run off forms a bath on which the dry dross floats; the operation is continued until sufficient dry dross has accumulated on the lead, to be raked out through the furnace door. The dross directly in contact with the lead is not touched till toward the end. When the level of the lead comes near the furnace door, enough is tapped into an outside kettle to leave a bath of lead for the charge next following to float on; charging, heating, and drossing are continued until a batch of dross is worked up. The temperature is always kept low, in order that no dross may be taken up again by the lead. The liquated dross is smelted in the blast furnace with a sulphur-bearing ore or by-product (galena, matte low in copper) to form a matte, or to raise the grade of the matte that has been added to the charge. The lead goes to the softening furnace.

The treatment of skimming aims to separate metallic lead, to desilverize

[1] Hofman, "General Metallurgy," 1918, 335.

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the skimming, and to collect the Cu in a matte. This is accomplished by smelting in a water-jacketed reverberatory furnace with the addition of high-grade galena (80 per cent. Pb). The metallic lead which had been withdrawn from the softening furnace as well as the lead reduced from the skimming by galena and fine coal spread over the charge, will collect on the bottom of the hearth. It will contain nearly all the precious metal of the skimming, as the finely divided globules of reduced lead passing downward through the charge will carry with them the precious metal, as is the case in crucible silver-assay. On top of the lead will float the molten skimming freed from metallic lead and precious metal. The resulting lead bullion, which has taken up about 1.5 per cent. Sb, goes to the softening furnace. The matte, containing Cu 25, Pb 55, S 55 and Sb about 1.5 per cent., is treated by a concentration-smelting process or goes to the converter to be blown to blister copper. The refined skimming with 15–20 per cent. Sb, some As, 0.2 per cent. Cu, and 1 ± oz. Ag per ton goes to the hard-lead blast furnace.

In a small reverberatory furnace, 10 ft. 6 in. by 9 ft. 6 in., with fire-box, 4 ft. by 4 ft. 6 in., served by undergrate blast, charges weighing 635 lb. (350 lb. dross, 150 skimmings, 100 galena, 20 sand, 15 fine coke), are dropped from the hopper in the roof in quick succession until the hearth is filled, and then replenished as fast as they melt, so that the fusion takes place on a bath of lead. The surplus of lead is ladled out from the well; speiss, matte, and skimming are tapped into an overflow slag-pot. In 24 hr. 90 charges are worked, yielding 16,000 lb. of lead bullion, a small amount of speiss (Cu 50, Pb 17 per cent., Ag 117 and Au 2½ oz. per ton), 3500 lb. matte (Cu 35 to 40, Pb 30 per cent., Ag 100 and Au 0.3 oz. per ton), and 5000 lb. clean skimming (Pb 75 per cent., Ag 1 to 1½ oz. per ton). The reason that such a large amount of material can be smelted in a comparatively small furnace is that it is treated on a bath of lead, and that the furnace never becomes cool. The bullion goes back to the softening furnace. The matte is smelted in a cupelling furnace with some silicious material into copper bottoms, 60-per cent. copper matte, and slag, the bottoms collecting the gold. The furnace is similar to the one represented in Figs. 388–392. The test-ring, which is not water-jacketed, is 6 by 8 ft., and is lined with a mixture of raw and burned clay; the cavity is 6 in. deep. The slag runs off from the front into an iron pot and, when the test is full of metal and matte, the contents are tapped into molds, placed on a truck beneath the test, by drilling with an augur a hole into the breast. The tap-hole is again closed from the inside by inserting a clay stopper at the back through the tuyère-hole; the furnace is then ready for another charge. When the molds are cold, the bottoms are separated from the matte. The matte is brought forward in another cupelling furnace to metallic copper to be used in precipitating silver from silver sulphate solution, obtained in parting doré silver by means of sulphuric acid. For every 100 lb. metallic copper produced, 26 gal. of reduced oil are required, an air-injector being used.

A large furnace for doing similar work is shown in Figs. 627–630. Its leading features are similar to those of the softening furnace given in Figs. 517–524.

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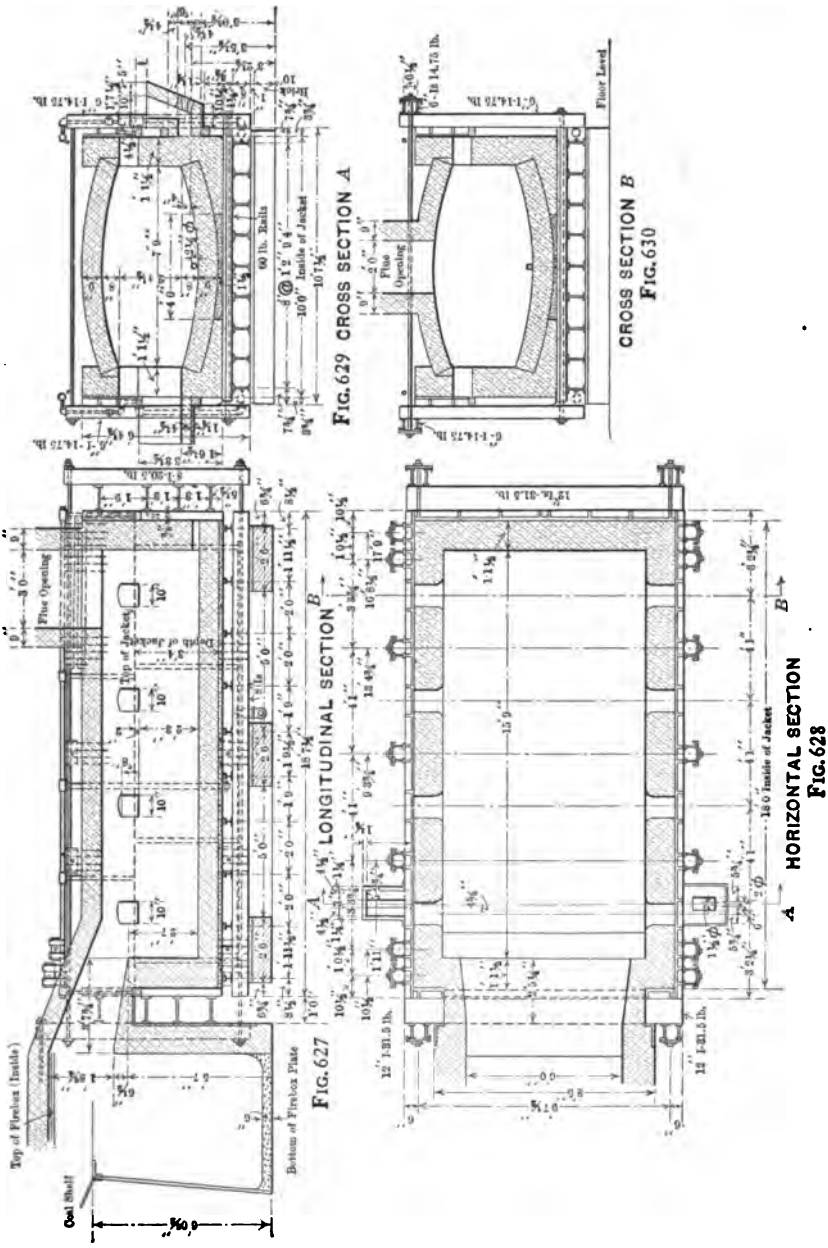
The fireplace is the same as the one illustrated in Figs. 592–594. The furnace rests on steel rails carried by longitudinal I-beams, is heavily ironed, and water-jacketed at sides and ends. It has at one side, the firebridge end, a lead-well



FIGS. 627 to 630.—Reverberatory furnace for antimony-skimming.

with tap-hole, and at the other a tap-hole for matte and skimming; at the flue-end is another tap-hole for emptying the furnace. The hearth is 15 ft. 9 in. by 7 ft. 9 in. and 2 ft. 3 in. deep, corresponding to 275 cu. ft., and a capacity

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for about 180 tons molten lead. The mode of operating is the same as with the smaller furnace; the amount of material treated is about 25 per cent. greater. At the works of the International Lead Refining Co., East Chicago, Ind., stibnite (Sb_2S_3) is frequently used as reducing agent instead of galena (PbS). A charge is made up of softening-furnace dross and skimming, litharge, and stibnite; it furnishes lead bullion (Pb 94 to 95, Sb 3 to 4, Cu 0.6 per cent.; Ag and Au 200 to 250 oz. per ton), matte (Pb 50 to 55, Sb 1.5 to 2.5, As 1, Cu 30 to 35 per cent.; Ag 100 to 150 and Au 0.06 oz. per ton), and refined skimming (Pb 45, Sb 20, As 1 to 3, Cu 0.25, SiO_2 6 per cent.; Ag 0.8 oz. per ton, Au none). It is surprising that the lead bullion does not run higher in Sb than that produced with galena as a reducing agent.

Smaller furnaces, with hearths 8 by 12 ft. and 10 in. deep and not provided with lead-well, sometimes have a tapping-slot with two tap-holes, the lower for lead and the upper for refined skimming, the lead to be collected in a kettle and the skimming in slag-pots. Other furnaces of similar size have a single tap-hole; the entire content of the furnace is tapped into a kettle with overflow-spout. While the lead is flowing into the kettle, part of it is ladled into molds if necessary. When the lead has run out, and the skimming appears, it will collect on the lead in the kettle and overflow into slag-pots placed beneath the spout. After the molten charge in the furnace has run out, the tap-hole is plugged, and new material fed into the furnace. The skimming floating on the lead in the kettle hardens in a short time, is raised to drain off adhering lead, and then removed. The lead in the kettle is cast into bars and goes to the softening furnace. When cakes of liquated skimming are rolled out from the slag-pots and broken, there will be found two products: liquated skimming assaying under 3 oz. Ag per ton, forming the bulk of the cone, and at the bottom a small cake of a silvery white antimonial speiss of Pb and Cu, which contains as much as 40 per cent. Cu and, with 250-oz. bullion, assays often as high as 300 oz. Ag per ton. Analyses of antimonial speiss are given in Table 143.

TABLE 143.—ANALYSES OF ANTIMONIAL SPEISS[1]

Pb	Sb	Cu	Fe	Zn	S
Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
79.96	16.62	0.327	0.157	0.271	0.026
78.67	18.22	0.860	0.142	0.251	0.026

Another sample gave Pb 63.3 per cent. and Sb 30.5 per cent., the antimonial lead produced assaying Sb 18 per cent. The speiss has a beautiful amethystine color, shows cleavage planes similar to those of spiegeleisen, and is sectile. Formerly it was considered preferable to tap the skimming or let it overflow on an iron plate, as it chilled quickly and was easily broken to pieces, but it ran higher in silver.

[1] Sample from A. W. Jenks; analyses by F. H. Davis.

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The liquated skimming is smelted in a small (36- to 42-in.) blast furnace, with about 100 per cent. of slag and 11 per cent. of fuel, care being taken to avoid a slag that is very ferruginous, as speiss might otherwise form.

The slag made at the International Plant[1] has the composition: SiO_2 24 to 26, Al_2O_3 up to 10, FeO 36 to 38, CaO 10 to 12, ZnO 12 to 14, Pb 1.5–2.0, Sb 0.5 per cent. With less Al_2O_3 in the mixture, a slag frequently made contains SiO_2 30, FeO 50, CaO 10 per cent.

In order to reduce the loss of antimony by volatilization (which is considerable), non-argentiferous galena may be added to the charge. The sulphur acts as a reducing agent, doing thus to some extent the work of C or CO. The amount of galena added is so regulated that no matte shall form; it is from 13 to 28 per cent. of the weight of the skimming. Though this addition reduces the percentage of antimony in the resulting hard lead, this is of no consequence as long as it contains from 15 to 18 per cent. Sb. At the works of the International Lead Refining Co., Hulst has added antimony ore, oxide as well as sulphide, to the blast furnace charge with most satisfactory results. The antimony oxide is reduced with the antimoniate of lead of the skimming; the sulphide reacts with the oxide, furnishing Sb and SO_2. According to Hering[2] the reaction $3Sb_2O_4+2Sb_2S_3=10Sb+6SO_2$ takes place in the presence of reducing agents. The volatilization loss inherent in the smelting of antimony ores in the blast furnace is taken care of by passing the gases through the bag house. In this manner a recovery of 97 per cent. Sb is readily obtained.

TABLE 144.—ANALYSES OF HARD LEAD AND HARD-LEAD DROSS

	Lautenthal		Clausthal	Přibram (*d*)	Pueblo Smelting and Refining Co. (*e*)
	Before poling (*a*)	After poling (*b*)	Dross from liquating hard lead (*c*)		
Sb	12.753	15.390	38.763	24.270	20.550
As	.....	.....	.....	0.109	0.100
Cu	1.861	0.152	37.643	0.160	0.300
Pb	85.291	84.650	22.962	74.886	79.042
Ag	0.0035	0.003	.....	0.009	0.008
Ni	.....	.....	(S 0.240)	0.015	.....
Fe	Trace	Trace	0.139	0.018	.....
Zn	Trace	Trace	0.232	0.009	.....
Sn	.....	.....	.....	0.524	.....

(*a*) Private notes. (*b*) *Ibid.* (*c*) *Berg. Hüttenm. Z.*, 1870, XXIX, 169. (*d*) *Oesterr. Jahrb.*, 1891, XXXIX, 64. (*e*) Howard, W. H., private notes, 1896.

The grade and grain of the hard lead are much improved by poling in a kettle for a few hours or liquating in a reverberatory furnace at a low temperature. The reason for this is that some of the Cu remaining in the softening skimming

[1] Hulst, *Tr. A. I. M. E.*, 1914, XLIX, 538.

[2] *Berg. Hüttenm. Z.*, 1883, XLII, 394.

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enters the hard lead on being smelted. The dross from liquating the crude hard lead has been found to contain as much as 40 per cent. Cu, but generally contains about 10 per cent. The hard lead is usually molded by ladling from the kettle in which it has been poled, or from the one into which it ran from the liquating furnace. Hard lead bars weigh 73 lb. The surface of the refined hard lead, when liquid, soon becomes covered with dross, and in order to obtain clean bars, without being forced to waste much lead by skimming, it is advisable to float a wrought-iron ring on the lead just large enough for the ladle to pass through, and to dip the lead from the ring. Thus only a very small surface will have to be kept bright. With large tonnages, hard lead is molded as is soft lead. Analyses of hard lead and hard lead dross are given in Table 144.

263. Tin Skimming.—Tin ore occurs so rarely with lead-silver ores that it is only an exceptional case when tin skimming is formed in softening lead bullion. At Freiberg, Saxony, the tin of the ores becomes concentrated to some extent in the lead bullion produced. The skimming that rises to the surface in the softening furnace after the dross has been removed has the following composition:[1] PbO 70.35, SnO_2 12.53, Sb_2O_5 12.50, As_2O_5 4.73, CuO 0.61 per cent.; and contains 72.9 oz. Ag per ton. Plattner has introduced a method of desilverizing this skimming and concentrating the tin in a marketable alloy which contains Sn 33, Sb 14, As 1 per cent.; the rest being lead. Details, with analyses of intermediary products, are given in the reference.

264. Kettle Dross.—The impurities in this dross are very few. It consists principally of lead oxides mixed with metallic lead, and is usually put back into the softening furnace with the next charge after the furnace dross has been drawn off. This is the best way of disposing of it.

265. Refining Skimmings and Polings.—An analysis of refining skimming gave Iles,[2] SiO_2 1.6, Fe 0.6, Zn 8.0 per cent., Cu trace, As none, Sb 4.02, Pb 75.0 per cent., Ag 1 oz. per ton, and Au trace. The percentage of Sb is exceptionally high. It is also singular that there should be present traces of Cu and Au. The skimming is best treated in a reverberatory furnace that is used for no other purpose, except perhaps for liquating hard lead, in which case the lead resulting from the refining skimming is sold as second-class lead, on account of the Cu it has taken up. If the reverberatory is used only for reducing refining skimming, the resulting lead can be worked in with the regular refining charges, and corroding lead produced. The hearth of a furnace suited for this purpose may be 10 by 5 ft., and 9 in. deep, built of fire-brick inclosed in a wrought-iron pan. It will have an inclination of 3 in. from the bridge to the flue, where the main working door is placed; a second door is placed in the middle of one of the sides to introduce and distribute the charge. The tap-hole below the flue discharges the lead into a small spherical kettle having a fireplace beneath. The English (§61) or Silesian (§62) reverberatory furnace also serves the purpose very well.

[1] Plattner, *Berg. Hüttenm. Z.*, 1883, XLII, 417.

[2] *School Min. Quart.*, 1899, XX, 397.

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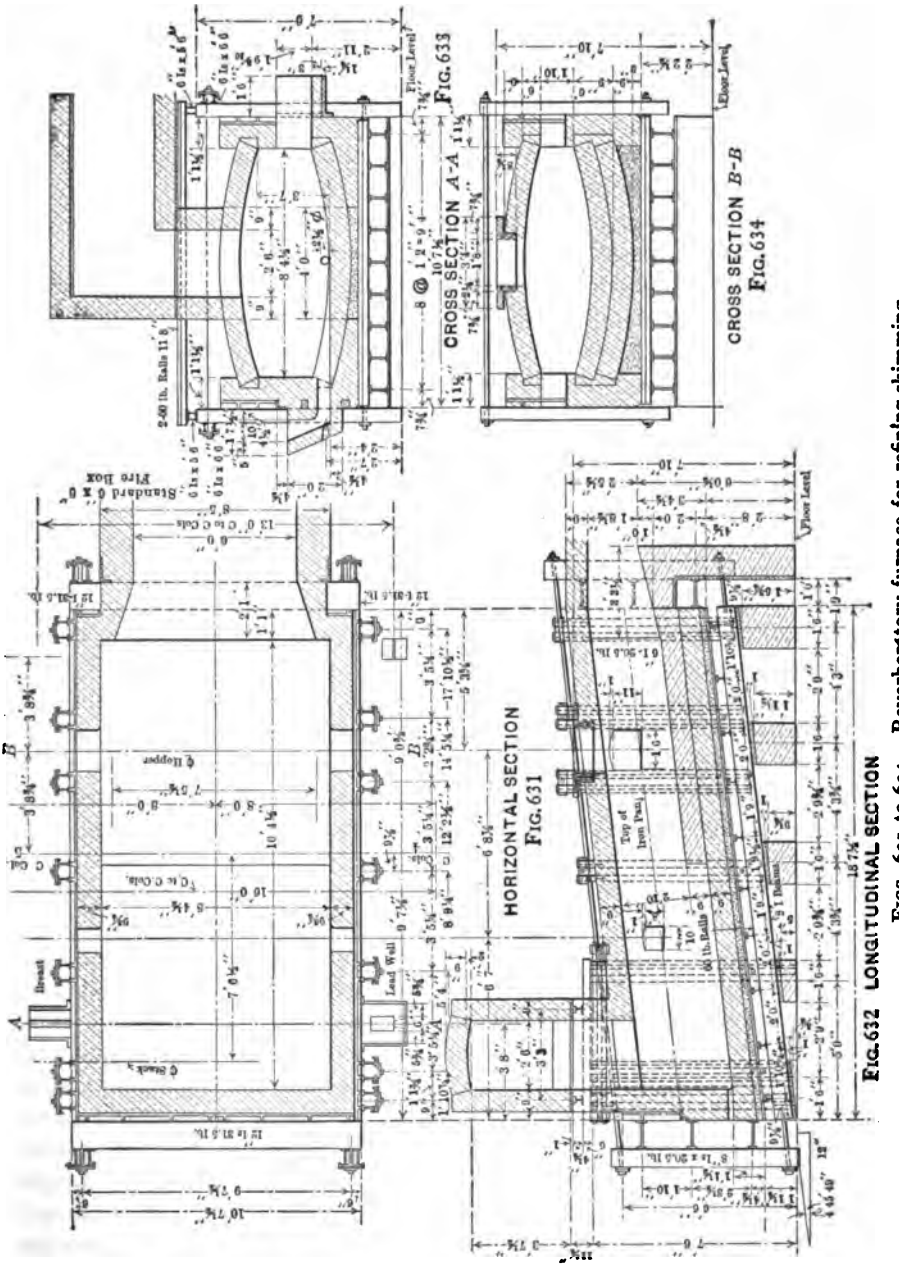

FIGS. 631 to 634.—Reverberatory furnace for refining-skimming.

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The mode of operation is as follows: The hearth of the furnace, dark red from a previous charge, is covered with a 2-in. layer of fine coal to protect it from the corroding effect of the skimming. It is then filled with skimming mixed with 10 per cent. of fine coal, leaving room for the gases to pass. Sometimes the charge reaches only to the working doors, being renewed from time to time as it shrinks. As the charge heats, lead will flow into the sump, whence it is tapped at intervals. The fire is gradually urged; when no more lead flows the charge is rabbled, and when exhausted, it is drawn.

Four tons of skimming, yielding about 60 per cent. lead, are worked in 12 hr., two men attending the furnace.

The residue, which contains Pb, Zn, Sb, perhaps some As, and coal ashes, is added to the smelting-charge of liquated softening skimming.

A large furnace, used in connection with the two 300-ton softening furnaces shown in Figs. 517–524, is represented in Figs. 631–634. The fireplace is the same as the one given in Figs. 592–594. The furnace slopes from firebridge to flue at an angle of 4° 45′ 49″, has a charging door in the roof near the upper end, two working doors at the sides, a lead-well with tap-hole, and a tap to empty the furnace. The hearth, 16 ft. 4½ in. by 8 ft. 4½ in., is built into an iron pan which rests on cross-rails supported by longitudinal I-beams; the latter are carried by cross-walls covered with iron plates. The bottom has two divisions: the lower, 9 in. thick, serves as a receiver for reduced lead; the upper, 18 in. thick, is the working floor for the charge. The mode of operating is similar to the one discussed in §254.

The refining skimming is sometimes charged into the softening furnace, after the dross has been removed, in order to assist in the oxidation of As and Sb. The former objection to this procedure, that zinc enters the hard lead when the skimming is melted in the blast furnace, has lost its importance since the present method of smelting in a reverberatory furnace with galena has come into use.

A. D. Divine[1] has devised a process for recovering as impure oxide the zinc of the refining skimming. The process, formerly in operation at the National Works of South Chicago, consists in crushing the skimming to ½-in. size, mixing with Na_2CO_3 and pulverized oil-coke (1 ton skimming, 175 lb. soda ash, 400 lb. coke), and smelting the mixture on a lead bath in an oil-fired reverberatory furnace, when Pb and Sb are taken up by the metal, while Zn, and with it some Pb, is volatilized, oxidized, and collected in a bag house. The charge is stirred at regular intervals through one door only in order to avoid cooling. When the Zn has been burnt off, the oil-burner is stopped, and the slag (soda skim) removed. The slag, which amounts to 10 to 15 per cent. of the weight of the charge, goes to the hard-lead blast furnace; the zinc oxide is treated electrolytically; and the lead forms a second-class product. In 24 hr. there are treated 18 tons charge. From 20,000 lb. refining skimming (Zn 16.50, Pb 63.50, Sb 6.00 per cent.) with 3300 lb. Zn, 13,700 lb. Pb, and 1200 lb. Sb, (or 14,900 lb. antimonial lead) there were produced 13,360 lb. antimonial lead and 3825 lb.

[1] *Tr. A. I. M. E.*, 1914, XLIX, 814.

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impure zinc oxide (Zn 65.00, Pb 14.65, Sb 1.00 per cent.), and 3320 lb. soda skim (Zn 24.50, Pb+Sb 9.8 per cent.). This accounts for all the zinc, and for 95 per cent. of the antimonial lead.

266. Rich Lead and Metallic Zinc.—The former goes to the cupelling furnace; the latter is used in desilverizing. It always contains a few ounces of silver.

267. Retort Dross and Blue Powder.—The retort dross is very rich in silver, which must be extracted quickly.

With the modern practice of pouring into an operating cupelling furnace the retort bullion with its floating dross, from the pot into which the content of a retort had been emptied, the question of treating the dross is settled quickly and efficiently. The zinc of the dross burns off readily, and impurities are quickly scorified by the litharge of the cupelling lead bullion.

In smaller plants this mode of operating is frequently not feasible. The dross, formed in small quantities, is worked off in the regular cupelling a little at a time, or at some works it is charged back into the retort. If there is too much for this, it is allowed to accumulate, and is scorified on the bath of lead free from or low in silver, with which a new test of the cupelling furnace is usually charged. Sometimes the retort dross is added to the softening furnace, after its own dross has been removed, that the silver may be taken up by the lead, and the impurities oxidized and taken up by the skimmings. It is not often that the dross is added to the charge of the bullion blast furnace.

The blue powder, consisting of a mixture of finely divided metallic zinc and zinc oxide, always contains some silver, say from 4 to 5 oz. per ton. It is not readily disposed of. If distilled by itself with the addition of charcoal, it yields from 33 to 55 per cent. of its zinc. At some works it is fed back to the retort with the following charge of zinc crust; at others it is added with the first zinc to the desilverizing kettle, that the metallic zinc may be taken up. If this is done, 50 lb. of blue powder are charged at a time into the kettle before the lead is tapped into it from the softening furnace. It is stirred into the bullion while this is rising in the kettle. It does not remove much silver from the kettle, but serves rather to saturate the lead with zinc, and to assist in removing the gold and copper. Sometimes the blue powder is screened to remove all coarse particles and is then sold to zinc works, or it is treated by zinc works which return to the refinery both the spelter and the residue. Since the treatment of mixed zinc-lead sulphide ores by a sulphatizing roast, followed by solution in sulphuric acid and electrodeposition of zinc, has become an established process, there is furnished a simple way of disposing of this unwelcome intermediary product.

In some European works it is dissolved in sulphuric acid and the zinc vitriol sold or converted into lithophone, a white paint which is a mixture of $BaSO_4$ and ZnS, resulting fron the double decomposition of BaS and $ZnSO_4$.[1] Since the use of the Howard stirrer and press, these products are being formed in small quantities only, and have lost the importance they formerly had.

[1] Webb, *Min. Eng. World.*, 1913, XXXVIII, 727.

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268. Litharge.—This is reduced in a reverberatory furnace or goes to the bullion blast furnace; if free from Cu, it may be added to the softening furnace after drossing. Though the reduction of litharge in the reverberatory furnace is preferable from a metallurgical and economic point of view, the blast furnace is in more common use, as refining works are thus enabled to smelt dry silver ores at a smaller charge than Western smelteries, which usually suffer from scarcity of lead ores. It is important for a refinery to smelt some ore in treating by-products in order to make new slag, and not be forced to smelt old slag over and over, which is expensive and causes losses in lead and silver, as these slags become gradually very rich in zinc.

269. Old retorts, cupel bottoms, etc., are added to the charges of the bullion blast furnace.

270. Results and Cost.—The output of silver with average lead bullion, running from 150 to 300 oz. Ag, and ± 0.5 oz. Au per ton, being pretty free from dross, but containing some As and Sb, is never less than 99.5 per cent.; generally there is a slight surplus of Ag, as the losses in the commercial assay are not taken into account in considering the silver that entered the plant. With Au, the yield is from 98 to 100 per cent.; a surplus is rare unless the lead bullion is very rich in gold. With Pb, the recovery is from 98 to 98.5 per cent. as long as the bullion contains little As and Sb; with bullion assaying As 2 and Sb 3 per cent., the lead loss will reach 4 and 5 per cent.

No detailed statement can be made about the cost of refining that would be generally applicable, as the single items vary greatly in different refineries. Iles[1] gives the operating expenses at Denver, Colo., in 1900 as $3.45 per ton of lead bullion, excluding reworking of intermediary products and overhead charges. At present this cost, including the reworking of intermediary products, ranges from $3.25 to $5.00 per ton, depending upon the size of plant and the facilities for handling materials. If overhead charges, such as interest, salaries, marketing, etc., are taken into account, the above figures will have to be doubled.

A good-size plant is one which treats about 5000 tons lead bullion per month. The works of the International Lead Refinery, East Chicago, Ind., which have an annual capacity of 60,000 tons lead bullion, are valued in the report of the Anaconda Copper Mining Co. at $480,000 (pre-war prices), which equals $8.00 per ton of lead bullion.[2]

271. Records.[3]—In a refinery where so many intermediary products result, it is essential to know how much silver, gold, and lead is contained in each of them. These quantities vary a great deal with the different kinds of bullion treated. It is therefore essential to be able to follow up the metals from the time they enter the refinery until they leave it. A form of the so-called "kettle book" is subjoined.

[1] *Eng. Min. J.*, 1900, LXX, 185.
[2] Ingalls, *Eng. Min. J.*, 1917, CIII, 1051.
[3] Argall, *Min. Sc. Press*, 1906, XCIII, 573, 722.

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KETTLE BOOK[1]

191– date		Lot	Name of bullion	Charge	Gross weight	Assay: oz. per ton		Content: total ounces	
Month	Day	No.		No.	Lb.	Ag	Au	Ag	Au

Softening dross	Softening skimming	Kettle dross	Net weight in kettle	Assay: oz. per ton		Content: total ounces	
Lb.	Lb.	Lb.	Lb.	Ag	Au	Ag	Au

Zinc added: lb. for	First crust	Assay: oz. per ton	Content: total ounces
Au, Ag	Lb.	Ag, Au	Ag, Au

Second crust	Refining skimming	Refined lead	Coal	Remarks
Lb.	Lb.	Lb.	Tons	

It is to be noted that the "net weight in kettle," before adding the first zinc, is found by deducting the sum of weights of softening dross, skimmings, and kettle dross from the lead bullion charged into the softening furnace. The "net weight in kettle," after adding the first zinc, is found in a similar way, by adding the pounds of zinc used for making the first crust to the net weight before the first zinc addition was made, and deducting the weight of the first crust from this sum.

The other principal books kept in a refinery are: the retort book; the

[1]Owing to the form of this volume, it has been necessary to put the headings in four rows one beneath the other, but in the actual kettle-book they simply run across the two opposite pages in one line.

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cupellation-book; the books for reverberatory furnaces I (liquating softening-dross and-skimming), II (reducing refining-skimming and liquating hard lead), and III (smelting dross and skimming with galena); and special assay books for the softening furnace, desilverizing kettle, retorts, cupelling furnaces, etc.

272. Relative Advantages of Parkes and Pattinson Processes.—The many advantages[1] that the Parkes process has over that of Pattinson have made it *the* desilverizing process used in the United States. There is, however, one instance in which Pattinson might be preferred to Parkes. It is when the lead bullion is rich in Bi. During the crystallization, Bi follows the liquid lead. It is thus concentrated in the rich lead, and can be recovered when this is being cupelled. The Bi content of the market lead thus becomes very much lowered, although not quite removed. In Parkes process, the Bi enters only to a small extent into the zinc crust, with the result that the refined lead may become richer in Bi than the original bullion. This difference is shown by the two analyses in Table 135 by Hampe[2] from Lautenthal market lead, where Parkes' process replaced that of Pattinson.

TABLE 145.—ANALYSES OF PATTINSON AND PARKES REFINED LEAD, LAUTENTHAL

	Pb	Cu	Sb	As	Bi	Ag	Fe	Zn	Ni
Pattinson..	99.966200	0.015000	0.010000	None	0.000600	0.002200	0.004000	0.001000	0.001000
Parkes....	99.983139	0.001413	0.005698	None	0.005487	0.000460	0.002289	0.000834	0.000680

An interesting combination of the two processes is found at Freiberg, Saxony,[3] where the original lead bullion, containing from 116 to 233 oz. Ag per ton, and from 0.02 to 0.06, rarely 0.16 per cent. Bi, is concentrated by crystallization to a liquid lead with 0.17 per cent. Bi, assaying 583 oz. Ag per ton, which is cupelled, while the crystals averaging 30 oz. Ag per ton are desilverized with zinc, the bismuth in the market lead not exceeding 0.02 per cent.

Large shipments of lead bullion with considerable quantities of bismuth are of rare occurrence in refining works. Occasional ones are worked in with other bullion that is free from, or at least low in, bismuth.

The practice of Freiberg has been reversed in the United States where the lead bullion usually shows higher values than that treated in Europe. The lead bullion is desilverized by means of the Parkes process, and the Bi-content of the refined lead lowered by Pattinsonizing to 0.05 per cent., the dividing line between corroding and non-corroding lead.

At the International Lead Refinery of East Chicago, Ind.,[4] refined Parkes

[1] Philips, *Eng. Min. J.*, 1887, XLIII, 363.
[2] *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1870, XVIII, 195.
[3] *Eng. Min. J.*, 1886, XLII, 400; *Berg. Hüttenm. Z.*, 1887, XLVI, 45, 192; 1889, XLVIII, 116.
[4] Hulst, *Tr. A. I. M. E.*, 1914, XLIX, 532.

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lead is likely to contain from 0.08 to 0.12 per cent. Bi; this is reduced by two crystallizations to 0.05 per cent. or less, or by a single crystallization if the crystals are pressed.

At the Omaha plant of the American Smelting and Refining Co., there was in operation the Tredinnick process (§213–218), which also produced corroding lead from lead which had been freed from precious metal by the Parkes process.

CUPELLATION PROCESS

273. Cupellation, General.—The process of cupellation has for its object the separation of lead and silver. It consists in melting and heating in a reverberatory furnace argentiferous lead with access of air, to the temperature at which litharge forms freely on its surface. This is run off and is in part absorbed by the hearth, while the silver, having scarcely any affinity for oxygen, remains behind in the metallic state. The oxidation of the lead is principally effected through the action of the blast playing over its surface, but is also assisted by the litharge when formed, as this absorbs oxygen and gives it off again to the lead and its impurities. The most important of these are Cu, As, Sb, Bi, Te, Ag, and Au.

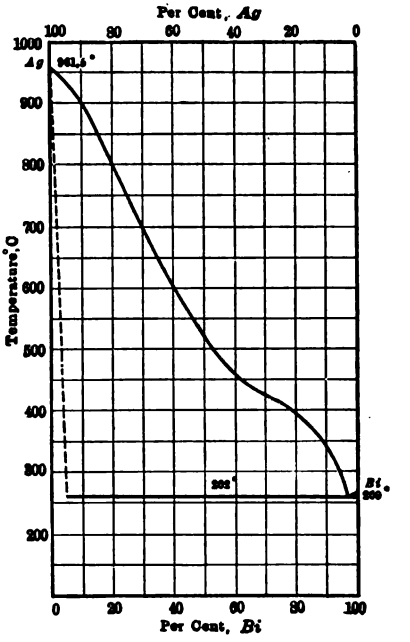

FIG. 635.—Alloy series Ag-Bi.

1. *Copper.*—The bulk of the Cu is removed with the dross. The Cu remaining with the Pb after drossing, is taken up only very gradually by the litharge. As it has less affinity for oxygen than Pb, its oxidation must be caused not so much by the action of the air as by a large quantity of litharge acting on a small amount of copper. With reversed conditions Cu_2O[1] oxidizes Pb. The oxidizing action of Cu_2O contained in the litharge seems to be the cause of the fact that cupriferous Pb is cupelled more quickly and with less loss in Pb and Ag than if the Pb were free from Cu. A satisfactory explanation for this is found in the fact that PbO and Cu_2O forms an eutectic (§29) which makes it possible to cupel at a lower temperature than would be the case if Cu were absent. Kerl[2] states that in cupelling 30 tons of argentiferous lead containing from ¾ to 1 per cent. Cu, the operation lasted 24 hr. less than when Cu was absent, and that the losses in metal were as 2:7.

[1] Berthier, "Traité des essais," Thomine, Paris, 1834, Vol. II, p. 663. Hofman, "Copper," 1918, 387.

[2] "Grundriss der Metallhüttenkunde," Felix, Leipsic, 1881, 270.

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2. *Arsenic and Antimony.*—These two metals are in part volatilized, in part oxidized and scorified by litharge, forming skimmings just as in the softening of lead bullion. They have no special influence on cupellation proper.

3. *Bismuth.*—Bismuth is concentrated with the Ag in the Pb until toward the end of the process, and then greatly retards the progress of the work. It is finally oxidized and enters the litharge, giving this a greenish color, and is absorbed by the hearth material, while the Ag also retains some Bi with great tenacity. The freezing-point curve of the alloy series Ag–Bi, Fig. 635,[1] shows an eutectic with 97.5 per cent. Bi, freezing at 262° C., and a solid solution of Bi in Ag of 5 per cent. If the Bi is to be recovered, the Ag is concentrated only to a certain degree (say to 50 or 60 per cent.) in the Pb, which is then cupelled in a separate furnace; the saturated part of the hearth of this furnace and the litharge form the raw material for the extraction of bismuth in the wet way.

Tellurium.[2]—This has a strong affinity for Ag, similar to that of Cu. The research of Pellini and Quereigh[3] has shown the existence of two chemical compounds, the stable Ag_2Te (hessite), melting at 959° C., and the unstable AgTe, melting at 414°, which undergoes a change at 412°; and two eutectics, Ag–Ag_2Te with about 12 per cent. Te, freezing at 872° C., and Ag_2Te(AgTe)–Te with about 70 per cent. Te, freezing at 351°. The later work of Chikasigi and Saito[4] replaces AgTe by Ag_7Te_4. Copper[5] forms with Te the two compounds Cu_4Te_3 (620° C.) and Cu_2Te (856° C.). In cupelling retort bullion with 0.31 per cent. Te at Pertulosa, Italy, the products given in Table 146 were obtained.

TABLE 146.—PRODUCTS FROM CUPELLING LEAD BULLION CONTAINING TELLURIUM

Products	Te	Se	As	Sb	Pb	Ag	Au
First litharge	0.2170	.....	.....	.....	.....	.....	.....
Crude silver, while brightening.	0.0487	Trace	Trace	0.0197	0.1785	(*a*) 96.2711	0.6190
Second litharge	3.3437	Trace	Trace	0.0972	74.3440	1.1292	0.0008
Refined silver	0.0130	Trace	Trace	Trace	0.0038	98.4937	0.6340

Products	Cu	Bi	Fe	Zn	Ni, Co	Cupel bottom	O and loss
First litharge	.....	.....	.....	.....	.....	.....	.....
Crude silver, while brightening.	1.9332	0.0140	0.0113	0.0032	0.0013	.....	.....
Second litharge	5.9709	0.0538	0.0608	0.0211	0.0041	2.4580	12.5174
Refined silver	0.8454	0.0028	0.0073	Trace	Trace	.....	.....

(*a*) By difference.

[1] Petrenko, *Zt. anorg. Chem.*, 1906, L, 138.

[2] Heberlein, *Berg. Hüttenm. Z.*, 1895, LIV, 41; *Min. Ind.*, 1895, IV, 481.

[3] *Atti Reale Accademia dei Lincei*, 1910, XIX[2], 415, 445, through *J. Inst. Met.*, 1911, V, 317.

[4] *Mem. Coll. Sc. Kyoto Imperial Univ.*, 1916, I, 316; through *Chem. Abstracts*, 1917, XI, 447.

[5] Chikashige, *Zt. anorg. Chem.*, 1907, LIV, 54.

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A sample of the silver shown in the table, taken during brightening, had a bluish-gray color, was brittle, and showed a fine-grained fracture. In order to remove the impurities, some lead was added, giving the second litharge and refined silver. Of the Te contained in the crude silver, 61 per cent. entered the litharge, 1.4 per cent. remained in the refined silver, and the difference, 37.6 per cent., was volatilized or absorbed by the hearth material. The volatile character of Te was clearly shown by assays of flue-dust, samples from 70 to 90 ft. from the furnace being richer in Te than those close by.

5. *Silver.*—The litharge obtained in cupelling always contains some Ag, the amount increasing with the richness of the lead. The presence of Ag is due to particles of argentiferous lead carried mechanically by the litharge, and to Ag or some Ag_2O held in solution by the litharge. Stahl[1] finds that Ag_2O loses O at 130° C., and cannot therefore exist at a cupelling temperature of about 800° C. On the other hand, Wait[2] dissolved from litharge containing 2.94 per cent. Ag; by means of cold acetic acid, 10.47 per cent. of the Ag-content, and, when brought to a boil, as much as 18.67 and 19.25 per cent. of the Ag; when added boiling, no Ag went into solution. As Ag is insoluble in the acid, the silver must have been present as an oxide unless there is formed a silver plumbite about which nothing is known. Kohlmeyer,[3] who investigated the subject, came to no conclusion as to the form in which Ag is present; he found, however, that molten litharge could dissolve from 3 to 6 per cent. Ag, and that its freezing-point was lowered by the Ag to 840° C. Dudley[4] finds that the dissolving power of litharge at 1100° C. for Ag is 7 times as great as that at its melting-point. His experiments show that litharge in contact with lead bullion will dissolve Ag until the distribution equilibrium for a given temperature has been established. If the litharge was richer in Ag than the bullion, Ag will be taken up by the bullion until the equilibrium has been reached. He finds that the concentration of Ag in litharge is directly proportional to its concentration in the bullion; that for bullion with up to 8 or 10 per cent. Ag, the concentration of the Ag in the litharge is proportional to the weight-percentage in the bullion; and that with richer bullion the proportionality lies between the atomic or molecular percentages of Ag in litharge and bullion. As regards the form in which Ag is present in molten litharge, he holds that, being a good conductor of electricity, molten litharge ought to show some degree of dissociation, and that the Ag, present in small quantities, ought to be more or less dissociated and be present for the most part as Ag ions.

6. *Gold.*—This follows the Ag in the cupellation, but none of it, or perhaps a trace, is found in the litharge.

According to the general construction of the furnace, and the consequent mode of operating, cupellation is generally discussed under two heads: German Cupellation and English Cupellation.

[1] *Metallurgie*, 1907, IV, 688.
[2] *Tr. A. I. M. E.*, 1886–87, XV, 463.
[3] *Chem. Z.*, 1912, XXXVI, 1079.
[4] *Met. Chem. Eng.*, 1916, XIV, 636, 695.

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274. German Cupellation, General.[1]—The characteristics of this method are: that a large reverberatory furnace with a fixed bed and a movable roof; that the bullion to be cupelled is all charged at once; and that the silver is not refined in the same furnace in which the cupellation was carried on.

275. Furnace.—The furnace selected for illustration is the one in operation at Přibram[2] (Figs. 636–639). It differs somewhat from the generally accepted circular form,[3] and is an improvement on it. Fig. 636 shows the fireplace, *a*, at the right, and the flue, *b*, at the left, of the hearth. Fig. 637 represents a horizontal section of one furnace, and the fireplace, *a'*, of a second furnace, the furnaces being built in pairs. The products of combustion pass downward through four separate flues, *b*, which unite in one main flue, *c*, leading to the dust-chambers. In Figs. 638 and 639 are seen two vertical sections on the same line, *EF*, Fig. 638 representing the furnace before, Fig. 639 after, tamping in the hearth. The furnace is built of common red brick, with the exception of the parts that are exposed to the flame, as indicated by the cross-hatching. In the upper part of the foundation and in the side walls small channels, *d*, and, *e*, reaching outward, are left open. They serve as drains for the moisture. At the back of the furnace are three openings, *f*, through which the tuyère-pipes are introduced. At the front is the litharge channel, *g*, which can be closed by a sliding door; a cast-iron breastplate, *h*, serves as support for the upper hearth, *i*. The movable arched roof, *k*, rests on an angle-shaped cast-iron ring, *l*, and is removed with a differential pulley suspended from a traveler. On the foundation is built a brick bottom, *m*, the brick being set dry. Beneath its lowest point, just below the cavity, *n*, is a cast-iron plate, *o*, to prevent any leakage of metal through the drain, *d*, should the working hearth crack or be injured in any other way. The flues leading from the fireplace, *a*, to the hearth are shown in *j*. The fire-brick part of the furnace is encased in cast-iron plates that have openings corresponding to the upper drains. The whole furnace is well bound together by buckstays and tie-rods. The fuel used is a mixture of bituminous coal and lignite; the ash-pit is closed, as undergrate blast is used.

276. Plattner Cupelling Furnace.—In this connection may be mentioned the Plattner[4] modified German cupelling furnace. It has the form of a reverberatory furnace; the hearth is rectangular in plan (13 by 8½ ft.), and receives its blast from two pipes on either side of the firebridge; the litharge channel

[1] The author is fully aware that the German cupelling furnace is hardly found in this country, and has been given up in many European continental plants. If, nevertheless, the subject is treated here in more detail than may seem necessary, the reason is that the steps common to both types of furnaces can be made clearer with the German furnace, as here the entire bullion to be cupelled is charged at the start and not gradually as with the English furnace. Thus, the several stages of the process follow one another in their regular order, and do not overlap and become obscured as is the case with the English method of working.

[2] *Oesterr. Jahrb.*, 1890, XXXVIII, 46; Privates notes, 1890.

[3] *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1874, XXII, 89; *Berg. Hüttenm. Z.*, 1872, XXXI, 415.

[4] Drawings in Arche, "Die Gewinnung der Metalle," Leipsic, 1888, Heft 1, Plate 1.
Schnabel-Louis, "Handbook of Metallurgy," 1905, 1, 705.
Results in *Berg. Hüttenm. Z.*, 1886, XLV, 211.

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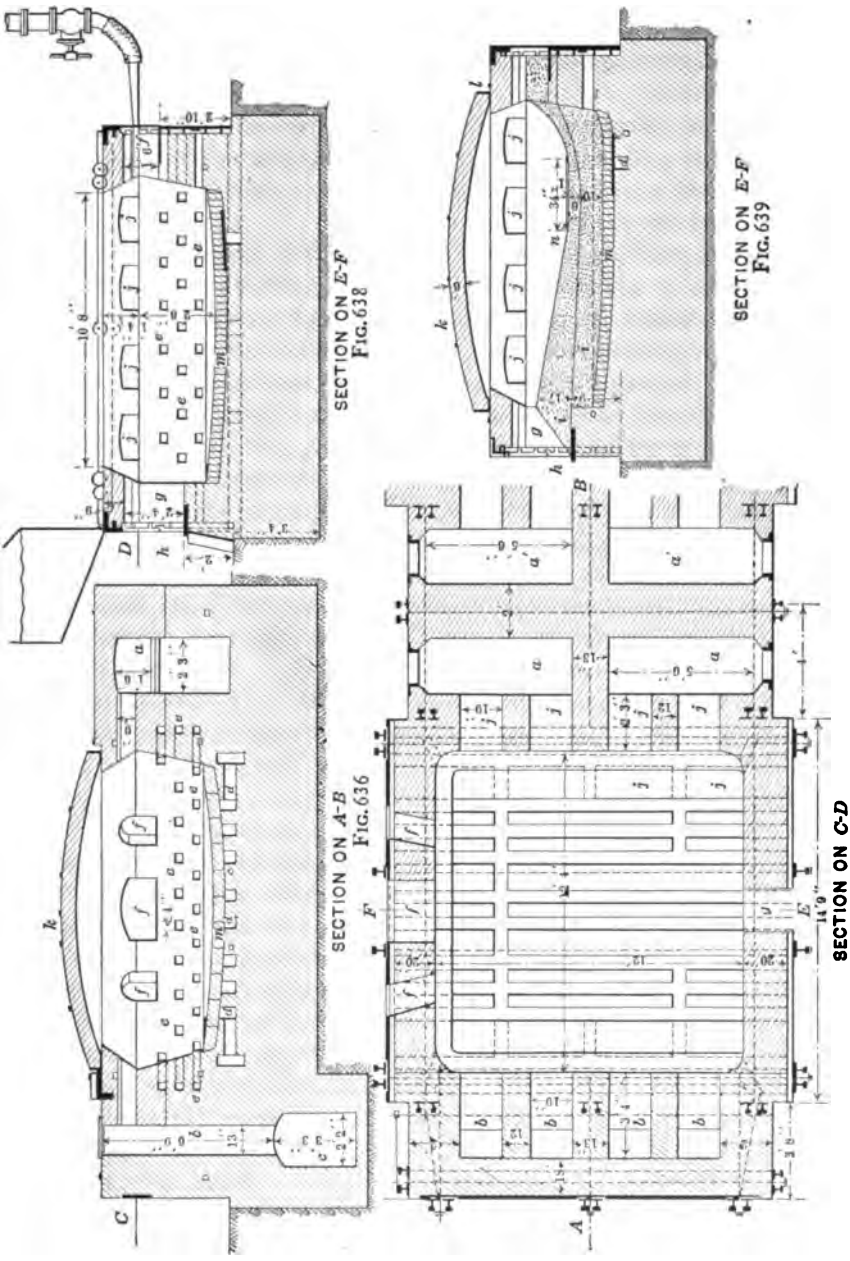

FIGS. 636 to 639.—German cupelling furnace, Přibram, Bohemia.

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is at the opposite end, beneath the flue, which carries off the products of combustion as well as the lead fumes. Less fuel is required and less metal volatilized, because the litharge need not be heated to such a degree to remain liquid; and no fumes enter the cupelling room, as they are all carried off with the fuel gases. Most of the English cupelling furnaces erected at present are constructed on the Plattner model.

277. Mode of Operating.—The operations necessary to work a charge are six in number: preparing the working bottom, charging and firing the furnace, softening the bullion, cupelling the softened bullion, removing the crude silver, and refining the crude silver.

1. *Preparing the Working Bottom.*—The material to be suited for preparing the hearth, must be basic, must not be attacked by the litharge, nor crack, must be sufficiently porous to absorb some litharge, and free from any reducing agent (organic matter, metallic sulphide). The material most used is a marl. The composition of that used in the Harz Mountains varies, according to Kerl and Wimmer,[1] within the following limits: SiO_2 21.22 to 22.24, Al_2O_3 5.39 to 6.76, Fe_2O_3 3.54 to 5.39, $CaCO_3$ 65.65 to 66.41, $MgCO_3$ 1.05 to 2.22 per cent. Usually the hearth material is a mixture of dolomite or limestone with fire-clay. For instance, at Tarnowitz[2] a dolomite of the composition, SiO_2 6.00, Al_2O_3 7.00, Fe_2O_3 4.10, $CaCO_3$ 49.86, $MgCO_3$ 32.82, is mixed with 25 per cent. of clay. At Freiberg[3] 2 vol. blue marl, ($CaCO_3$ 57.0, $MgCO_3$ 2.0, $Fe(Al)_2O_3$ 1.9; clay-substance 38 per cent.) and 1 yellow marl ($CaCO_3$ 54.53, $MgCO_3$ 20.55, SiO_2 12.72, Al_2O_3 6.07, $FeCO_3$ 1.77, H_2O 4.50 per cent.) are crushed in a ball mill with 10 per cent. clay (Al_2O_3 30.44, SiO_2 58.28, MgO 0.08, CaO 0.26, Fe_2O_3 0.90, K_2O 0.06, per cent.) to suitable size. At Přibram three parts of limestone are ground together with one part of clay so as to pass a 5-mesh sieve; at other places an 8-mesh sieve is used. The hearth material has to be moistened before it is beaten down in the furnace. For this purpose, it is spread on the floor, sprinkled with water from a hose, and turned over and over with a shovel, that the moisture may be equally disseminated through the powder. If left over night, it must be covered with wet cloths and worked again the next day. The material is of the right consistency if, when pressed in the hand, it coheres to a lump, but has not sufficient moisture to adhere to the hand. The hearth material is sifted through a coarse hand-sieve, just before using, and any lumps that may have formed are broken up or thrown aside. Sometimes it is introduced all at once, sometimes in two separate layers. The latter is the way at Přibram, as Fig. 639 shows, *i*, being the upper and, *i'*, the lower bottom. Before the prepared mixture is put in, the brick bottom is sprinkled with water, that it may not take up any moisture from the hearth material. At Přibram the lower bottom, *i'*, is first tamped down to the form shown in the drawing. The tool required, the tamping-iron, is a cast-iron disk of about 6 in. in diameter and 1 in. in thickness, with a socket into which

[1] *Berg. Hüttenm. Z.*, 1853, XII, 241.
[2] *Zt. Berg. Hütten. Sal. W. i. Pr.*, 1884, XXXII, 107.
[3] Kochinke, *Freiberg. Jahrb.*, 1898, 98.

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fits a wooden handle about 4 ft. long. The tamping is begun at the center, proceeds in the form of a spiral to the side walls, and returns in the same manner, care being taken that the circular indentations shall overlap in part those made in working outward. By giving attention to this point, the surface will be beaten evenly, which is essential. Before putting down the upper bottom, the surface of the lower one is roughened by scratching with the point of a chisel. This is done that the bottom to be put down may adhere to the one already in place. The tamping of the upper bottom is done in the same way as the lower, only the surface, when finished, must be perfectly smooth. It is of prime importance that the hearth should have just the right degree of hardness. This is easily indicated to the ear and hand after a little experience. If too hard, it will crack and not be sufficiently porous; if not hard enough, it will absorb too much lead. If the material was too dry, the hearth will peel when heated; if too wet, it cannot be beaten to the desired hardness, as it acts like a cushion. In order to prevent its adhering to the tamping-irons, these are slightly warmed. The thickness of the hearth at the bottom and sides varies somewhat; the least is perhaps 6 in. at the bottom and 8 in. at the sides. The general rule for the curvature of the hearth is that the more concave the bed is the easier will be the cupelling and the harder the finishing; the flatter the bed, the harder the cupellation and the easier the finishing. When the hearth is completed, a cavity, *n* (1 in. deep and 34 in. in diameter), is cut in the deepest part to receive the silver. It is situated a little to one side of the medial line, toward the fire-bridge, that the silver may easily be kept molten at the end of the operation.

2. *Charging and Firing the Furnace.*—The furnace receives its charge of 25 tons of lead, enriched by the Luce and Rozan process. Sometimes the bottom is covered with straw before charging, to prevent its being damaged during the operation. The bars are placed in such a way as to leave an open space reaching from the tuyères to the litharge channel. The hood is then lowered on a clay lute placed on top of the furnace. The litharge channel is closed by lowering the door, the fire kindled on the grate, and soon the blast below let on.

3. *Softening the Bullion.*—The lead melts down slowly. The dross rises to the surface and is drawn off through the litharge channel. The temperature is raised and the blast put on through three tuyères, the skimming forms and is drawn off, and finally pure litharge takes its place. At the very beginning, when the dross is being removed, bubbles of gas are seen to pass off from the surface of the lead, and this continues through almost the whole cupellation. They are mainly H_2O-vapor and are mixed with CO_2 resulting from the decomposition of the magnesium and calcium carbonates of the hearth material.

4. *Cupelling the Softened Bullion.*—The temperature is lowered and the volume of blast diminished. Both are kept low during the larger part of the operation; they are raised only toward the end, when the enriched silver-lead alloy requires a higher temperature to give up the last parts of lead. When the skimmings have been removed and the cupellation has somewhat progressed, the

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convex surface of the lead will be exposed to the action of the blast, while the lead near the periphery will be covered with litharge. The width of this rim depends on the rate at which the litharge is allowed to run off through the litharge channel. As litharge melts at 883° C., a temperature of 900° C. has to be maintained, if it is to remain liquid. Lead, melting at 327° C. would be volatilized to a considerable extent, if fully exposed to the action of the blast at this high temperature. The litharge is therefore allowed to run off only to such an extent as to give the rim a width varying from 12 to 15 in. at the beginning, and of 5 in. toward the end of the operation. The blast playing on the surface of the lead, forms small waves and drives the litharge toward the channel. The pressure is about 8 oz. per sq. in., and about 300 cu.ft. of air are delivered per min. through the three tuyères. In order to remove the litharge, a gutter is cut into the channel by means of a scraper. This is an iron rod, 8 ft. long and ⅜ in. in diameter, flattened out at both ends. One of these is bent to encircle a wooden handle, and the other, only slightly flattened, is sharpened and bent to the form of a hook. In cutting the gutter, the entire edge of the tool must be used, and not one of the corners. If larger pieces of the breast are to be cut out, it is done with a chisel-pointed bar, say ¾ in. in diameter. The rate at which the litharge runs off depends on the depth of the gutter and the strength of the blast. The depth is correct when the litharge runs off in a thin stream which stops as soon as the blast is lowered. If it runs too fast, the rim of litharge in the furnace decreases, and lead is volatilized; the litharge is not sufficiently desilverized, from too short a contact with the lead beneath it. If it runs too slowly, the rim of litharge becomes too broad and the cupellation is retarded; there is again loss in silver by the higher temperature that is required to keep the larger amount of litharge fluid, and if the temperature be not raised sufficiently, lead will be carried out mechanically by the litharge. The litharge gutter is first cut into the breast on the side farthest away from the fire-bridge and gradually moved toward the opposite side, that the final litharge may be drawn off as near the fire-bridge as possible. The current of the litharge, when flowing out of the furnace, is directed in such a way as to form a large cake in front of the furnace. In some instances a U-shaped piece of sheet-iron is placed upright in front of the channel, that the litharge may collect in a rectangular block. Before removing this, the litharge in the center, that is still liquid, is tapped from near the bottom of the cake. The litharge resulting from the cupellation is graded according to the silver contents and the percentage of impurities (copper, bismuth, etc.), into marketable litharge and into a by-product to be treated by a separate process. As the cupellation progresses toward the end, the temperature is raised and the blast increased, the side tuyère-openings are closed, and two tuyère-pipes introduced through the central opening; finally, the last film of litharge disappears from the surface of the lead with a characteristic phenomenon, the so-called brightening,[1] which every reader has watched while making a silver assay.

[1] Van Riemsdijk, *Berg. Hüttenm. Z.*, 1880, XXXIX, 247, 275, and Bock, *op. cit.*, 1880, XXXIX, 409, have made interesting investigations on this subject.

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The products of the Přibram cupelling furnace have, according to Dietrich, the compositions shown in Tables 147 and 148.

TABLE 147.—CUPELLATION PRODUCTS OF PŘIBRAM

	Dross	Tin skimming	Antimony skimming	Red litharge	Green litharge	Cupel bottom	Flue-dust
Pb	30.75	13.40	.....	.....	.....	.....	.....
PbO	55.27	64.97	77.95	98.370	98.140	68.860	64.41
PbO combined with $(AsSb)_2O_5$	.....	.....	11.87	.....	.....	.....	.....
CuO	1.99	0.29	0.28	0.069	0.080	0.070	.....
Bi_2O_3	.....	.....	.....	.....	.....	Trace	Trace
MoO_4	.....	.....	.....	.....	.....	Trace	.....
As_2O_5	1.42	1.87	0.92	0.010	0.009	.....	Trace
Sb_2O_5	1.83	6.76	5.85	0.074	0.067	0.530	11.40
SnO_2	0.72	10.31	Trace	.....	.....	.....	.....
Ag	.....	.....	0.004	.....	.....	.....	.....
Ag_2O	0.307	0.189	.....	0.004	0.0048	0.170	0.013
Al_2O_3	0.54	0.23	0.32	0.072	0.056	2.120	.....
Fe_2O_3			0.14	0.010	0.014	0.300	.....
ZnO	0.13	0.05	Trace	0.009	0.012	Trace	0.50
Ni	0.09	Trace	.....	.....	.....	.....	.....
NiO	.....	.....	0.04	0.005	0.005	.....	.....
CaO	0.45	0.43	0.95	0.256	0.362	.....	Trace
$CaCO_3$	.....	.....	.....	.....	.....	24.100	.....
MgO	.....	.....	.....	.....	.....	Trace	.....
CO_2	.....	.....	.....	0.383	0.432	.....	.....
SiO_2	1.75	0.94	0.37	0.320	0.350	2.970	4.35 +ash
S	2.30	0.65	0.09	.....	.....	.....	.....
SO_3	4.12	1.37	.....	0.034	0.027	0.040	16.65
C	.....	.....	Trace	.....	.....	.....	Trace

TABLE 148.—ANALYSES OF CRUDE SILVER

	Přibram (*a*)	Freiberg (*b*)	Wyandotte, Mich. (*c*)	
Ag	95	92.180	98.6910	99.5930
Pb	5	4.210	1.0900	0.2600
Cu	.....	2.104	0.1170	0.1060
Ni, Co	.....	0.600	0.0040	0.0080
Fe	.....	.....	0.0900	0.0310
Bi	Trace	.....	0.0058	.....
Au	Trace	.....	0.0023	0.0015

(*a*) *Oest. Jahrb.*, 1890, XXXVIII, 40. (*b*) Stölzel "Metallurgie," Vieweg, Brunswick, 1863–86, 1182. (*c*) *Tr. A. I. M. E.*, II, 97.

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The time required to cupel the 25 tons of rich lead is 80 hr. It is divided as shown in Table 149.

TABLE 149.—TIME OF CUPELLING CHARGE, PŘIBRAM

	Hours
Preparing the hearth material, making the hearth, and charging the lead.	8
Melting, and wheeling the necessary coal	16
Drossing	6
Drawing the tin skimming	6
Drawing the antimony skimming	3
Running off market litharge	23
Running off rich litharge	18
	80

The cupellation is in charge of three men, each with a helper, working in 8-hr. shifts. For every 100 tons of lead bullion are consumed 19.63 tons of coal and 23 bushels of hearth material (which includes the refining of the silver). The loss in Ag is 0.83 per cent.; that in Pb, 4.33 per cent. The figures do not include the loss endured in re-treating some of the by-products. Thirty-six per cent. of the litharge is low enough in silver to be sold in the market, the bullion from which it is made averaging 167 oz. Ag per ton.

5. *Removing the Crude Silver.*—After the silver has brightened, the blast is shut off, the tuyère-pipes are removed, and the litharge gutter is closed with a ball of clay. Two knife-shaped pieces of wrought iron are introduced through the litharge channel and pressed into the cake of crude silver. First warm, then cold, water is allowed to run into the furnace, and the silver then removed through the central opening at the back. It is cleaned, weighed, etc. The furnace is left to cool till the next day, when the hearth is examined for small particles of silver that have adhered to it. The upper hearth is then removed with a pick. Part of it is soaked with litharge to a depth of 2 or 3 in. This is separated from the unsoaked part, which is mixed in with the hearth-material for the next charge, while the lead-soaked part goes to the blast furnace.

6. *Refining the Crude Silver.*—The refining of the crude silver has for its object the removing of impurities, which vary from 2 to 10 per cent. Formerly this was done exclusively in a small oval reverberatory furnace, having a working door at one side or at the flue-end, and a working bottom of similar composition to the cupelling bottom, the reason being that the loss in silver and the consumption of fuel were considered very much smaller than if the firing was done in the big cupelling furnace. Ohl[1] and Foehr[2] have since proved this not to correspond to the facts, and the refining in a separate furnace has received a check. Since Roessler[3] discovered how to refine crude silver in a plumbago crucible by means of silver sulphate, the refining in a separate furnace has been abolished entirely at some works.

[1] *Berg. Hüttenm. Z.*, 1879, XXXVII, 274.
[2] *Ibid.*, 1885, XLIV, 381.
[3] *Ibid.*, 1889, XLVIII, 387.

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Refining in a crucible is carried on at small works by melting the silver, and uncovering the crucible that the air may oxidize the impurities. These are stiffened by sprinkling bone-ash or hearth material on the silver and then removing it with a skimmer, the operation being repeated till no more impurities rise to the surface. A slag obtained by Curtis[1] at Wyandotte, Mich., sand being used in refining, contained in addition to silicate of lead, the following metals: (NiCo)O 0.550.; CuO 0.203.; Bi_2O_3 0.026.; Ag 1.837.; Sb_2O_4 0.639.; As_2O_3 0.005.—total 3.260 per cent.

Roessler found that if Ag_2SO_4 is added to melted Ag in a crucible, first the Pb and then the Bi are converted to sulphates, the Ag being at the same time set free. Cu is not removed by Ag_2SO_4. By keeping separate the different slags he concentrates the Bi in a comparatively small amount of slag, to be treated separately, while the first slag contains most of the Pb. To prevent the crucible from being attacked, he introduces a layer of quartz-sand on top of the Ag, and then stirs in the Ag_2SO_4 in the middle. The sand serves at the same time to stiffen the slag, which is then removed with a skimmer. The process, as seen by the writer at the Lautenthal Smelting and Refining Works in 1890, differs slightly from the manner indicated above. It is as follows: The Ag_2SO_4 is produced by dissolving Ag in H_2SO_4, of 66° Bé., in a small cast-iron kettle. The solution is allowed to cool, is then diluted to 60° Bé., whereupon nearly all the Ag_2SO_4 will fall out as a slightly yellow cheesy mass. The supernatant liquor is drawn off as much as possible, and the remaining dilute acid driven off by heating. Special arrangements are required to cool the vapors, as they carry finely divided Ag_2SO_4 along with them. The temperature is raised to redness in order to fuse the Ag_2SO_4 (melting-point, 651° C.), which, when liquid, is cast into molds and is ready for use. The color of the melted sulphate is grayish green; it is hygroscopic, and is therefore kept in a lead-lined wooden box; 1000 parts contain 650 parts Ag.

Crude silver of a fineness varying from 950 to 980 thousandths is melted in a plumbago crucible holding 700 lb. The crucible is heated with coke in a small cylindrical furnace having in the lower part two 1¼-in. openings for the blast-pipes. On trying to stir in the Ag_2SO_4, as advised by Roessler, it was found that sometimes it got beneath the layer of sand, spread over the silver, and corroded the crucible. To prevent this, a wrought-iron ring (⅛ in. thick, 10½ in. in diameter, and 7 in. high) is coated on either side with a 3-in. layer of clay, and placed on the silver. Into the center are introduced with a ladle from 6 to 8 lb. of sulphate (the size of a hen's egg) that has been warmed. As soon as it comes in contact with the silver this begins to boil. When the effect decreases, the silver is stirred with an iron rod to assist the action of the sulphate. From 25 to 30 min. after the sulphate has been added, this is completely decomposed, a slag has collected on the surface of the silver, and quartz is added to stiffen it, that it may be removed with a skimmer. A second, a third, and, if necessary, a fourth addition of Ag_2SO_4 is given to make the silver fine. The test made for fineness is to dissolve some Ag in

[1] *Tr. A. I. M. E.*, 1873–74, II, 98.

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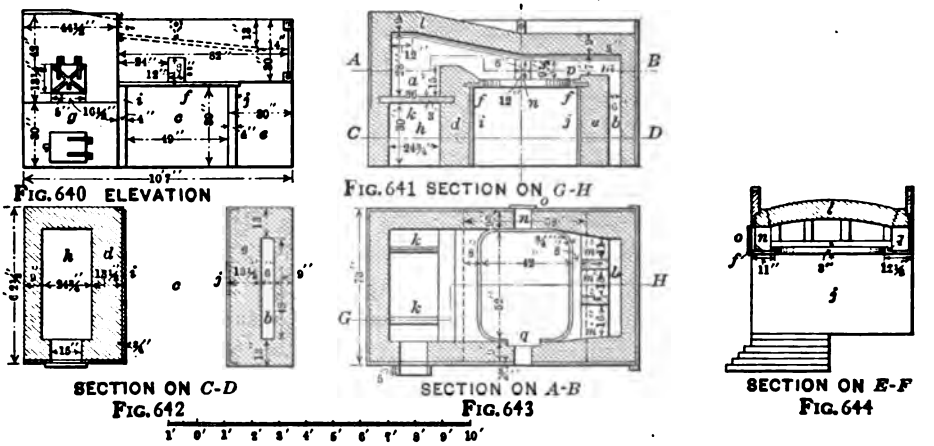
HNO_3 and to supersaturate with NH_4O. No precipitate must form even after standing.

The amount of Ag_2SO_4 required to fine the Ag is about 1½ times the total quantity of base metal present. Thus 700 lb. of crude Ag, being 970 fine, contain 21 lb. of base metal, which would require 31 lb. of Ag_2SO_4 to be added in three portions. If the test with NH_4O should prove this not to be sufficient, an extra addition is made. In 1890, 107,031 lb. of crude Ag with an average fineness of 970 required 6009 lb. of Ag_2SO_4 which corresponds to about 2 parts of Ag_2SO_4 to 1 part of base metal. Not all the Ag of Ag_2SO_4 is taken up by the Ag in the crucible; part of it enters the slag, as shown by the following analysis made by Hampe:[1]

SiO_2 40.7, P_2O_5 0.64, SO_2 0.61, S 0.15, FeO 13.47, Al_2O_3 0.43, Bi_2O_3 6.01, PbO 33.50, Ag_2O 2.05 (=1.88 Ag.), Cu 0.45, Sb 0.02, CaO 1.73, MgO 0.25, K_2O 0.64, Na_2O 0.26 per cent.

The main advantage of Roessler's method of refining is to be found in the larger direct output of silver, and the concentration of the bismuth in a comparatively small amount of slag that is more easily worked than cupel-bottom and litharge obtained in the reverberatory used for refining the silver.

278. English Cupellation, General.—The characteristics of this method are, a small or large reverberatory furnace with a movable bed and a fixed roof, and the fact that the bullion to be cupelled is charged gradually, and the silver usually refined in the same furnace in which the cupellation was carried on.



FIGS. 640 to 644.—Old English cupelling furnace.

279. Furnace.—This has undergone many changes from the original English furnace as described by Percy.[2] Figs. 640 to 644 represent one form of cupelling furnace which was commonly used in American refining works. The vertical section (Fig. 641) shows the general construction of the furnace, with fireplace, *a*, vertical flue, *b*, and space, *c*, between firebridge wall, *d*, and

[1] *Berg. Hüttenm. Z.*, 1891, L, 187; *Eng. Min. J.*, 1891, LII, 571.

[2] "Metallurgy of Lead," 178.

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flue wall, *e*. This is closed at the top by compass-ring, *f*, and the test when it has been put in place. The upper part of the furnace is encased in cast-iron plates; the side castings have a strengthening rib to resist the thrust of the roof. In addition, the front, *g*, of ash-pit, *h*, and the flue-end of the furnace, as well as inner sides *i* and *j* are protected by castings. The usual buck-stays and tie-rods have been left out in the drawings. To be noted are the large grate area (4 ft. 6 in. by 2 ft. ¾ in.), in comparison with the hearth area (4 ft. 4 in. by 3 ft. 6 in.), the height from the grate bearer, *k*, to the roof, *l* (2 ft. 4 in.), and the short distance (9½ in.) between the roof and the top of the compass ring, as they are all essential for good working. The flame from the grate-bars is directed downward by the pitch of the roof, and, being forced to pass through the small space between the shallow roof and the hearth, exerts all its heating force on the lead. By paying special attention to this part of the construction, it is possible to obtain a sufficiently high temperature to refine silver without being forced to use special kinds of bituminous coal. In fact, with undergrate blast, slack coal is good enough for cupelling, and nut coal is required only for fining, the coal being of ordinary grade.

The grate in the drawing is so arranged that firing, as well as stoking, is done from the front. Another plan is to place the grate-bars parallel with the ends of the fireplace, and to stoke from the side of the furnace. If in such a case forced blast is used, the casting at the stoking side has two oblong openings, each to be closed by a cast-iron door, say 21 in. long and 6 in. high, having its two hinges on the lower side. The horizontal flue, *m* (Fig. 641), is seen in Figs. 643 and 644 to be divided into three smaller flues, *m'*, increasing in width (11, 13, 16 in.) from the back of the furnace toward the front, the object being to prevent the flame from taking the shortest line toward the center of the flue, and to draw it somewhat toward the front, whereby the litharge floating on the front part of the lead is kept hot. At some works this flue is divided into five or six smaller flues. If the conditions of the draft are such that the flame rushes too much toward the center, this is remedied by placing one or more fire-bricks in the flue, which will correct the evil. At the back of the furnace there is only one door, *n*, through which passes the blast-pipe and through which is fed one pig of lead at a time. Most furnaces have three small openings, a central one for the blast-pipe and two lateral, through each of which a pig of lead is gradually pushed forward and melted away. The compass-ring, whose inner contour must, of course, correspond to the form of the test that is to pass through it, is intended for a rectangular cast-iron test having rounded corners. The upper rim, *p*, serves to hold in place the brickwork, with which its surface is covered. It extends all around the central opening, with the exception of the front, where it stops for a distance of 16 in., leaving room for the slot, *g* (4 in. wide), through which the litharge is to run into the litharge-pot. The outer contour of the compass-ring has the rectangular form of the furnace, and reaches from front to back. It thus forms at the front the support for the working tools, and at the back the support for the prop which gives the blast-pipe the desired pitch. At the front, the compass-ring is often left open the

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entire length of the litharge-slot. In such a case the support for the tools is a special cross-bar (skimming-bar) let into the brickwork, or fastened in some other convenient manner.

This furnace with working door at the side has three disadvantages. The draft sucks in air through the working door and cools the litharge flowing from the furnace, hence a temperature higher than necessary for cupellation has to be maintained in order to keep the litharge gutter hot; the size of the furnace is unduly limited and with it the rate of cupelling; the directions of flame and blast being at right angles to one another, causes an irregular travel of the litharge, and with it a likelihood of pellets of bullion being carried off.

Since the advent of the Plattner modification (§265) of the German cupelling furnace, the Plattner principle of construction has been applied to the English furnace. Such a furnace is the one represented in Figs. 645–653, which was in operation at the works of the Pueblo Smelting & Refining Co. Pueblo, Colo., until the refining department was closed. The side elevation (Fig. 685) and plan (Fig. 686) show the general construction of the furnace. From the grate, *a*, worked with undergrate blast, the flame passes through a narrow flue, *b* (3 ft. wide), on to and across the hearth, *c* (5 ft. wide, increasing to 5 ft. 3½ in.), ascends in the vertical flue, *d* (2 ft. 6¾ in. wide), returns through the horizontal flue, *e* (23 in. square), and then descends through *f* (of same size) into a main flue underground. Air for cupelling is delivered on to the hearth by the water-cooled tuyères, *i* and *i'*, on either side of the firebridge. The brickwork of the fireplace is protected on the hearth side by a heavy casting, *g*, the other three sides are strengthened by horizontal rails, *h* and *h'*, laid in the brickwork and secured by buckstays and tie-rods. The side walls of the hearth are inclosed by a heavy casting, *j*, the flange of which, *q* (Fig. 645, 646 and 648), supports them and forms at the same time the compass-ring for the test. The back of the casting is let into the brickwork of the fireplace, its sides are supported by the buckstays, *k*, and its front by the legs, *l*. The flue, *e*, rests on iron plates, *m*, which are supported by longitudinal rails, *n*; these are carried by the transverse rails, *o*, which are bolted to the flanges of two pairs of buckstays, *k*, and by the brickwork of the flue, *f*. The flue is bound with angle-irons and tie-rods. The test (Figs. 645, 646, 647, 649 and 650) consists of an oval ring, *r*, of boiler-iron, strengthened at top and bottom by flanges, *s*, and bound at the front by a loop-shaped band of iron. It rests on a cast-iron bed-plate, *t*. Its manner of support is shown in Figs. 652 and 653. Two rails *AA'* run along the bottom of the test; at one end they rest on the rounded bar, *B*, supported by a pair of jackscrews, at the other they are held in place by the looped rods, *RR'*, suspended from the yoke, *Y*, which is connected with a second yoke, *Y'*, by means of the right- and left-hand screw *SS'*. By turning the wheel, *W*, the front of the test can be raised and lowered without difficulty. Into the loop at the front fits the cast-iron breast-jacket (Fig. 651), which rests on the cast-iron bed-plate, *t* (Figs. 645–647). The test is not water-cooled, but in order to protect the filling material from the corrosive action of the litharge, a pair of water-pipes (*w* and *w'*, Fig. 647) is imbedded in it.

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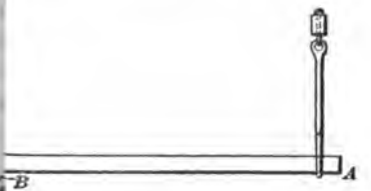

FIG. 652.—SIDE ELEVATION

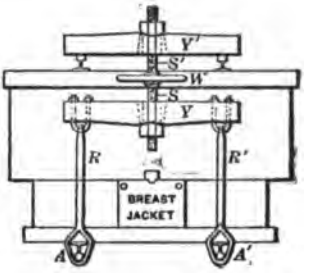

FIG. 653.—FRONT ELEVATION

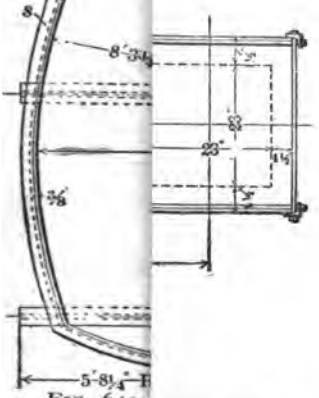

FIG. 649.

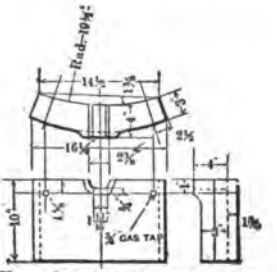

FIG. 651.—BREAST JACKET

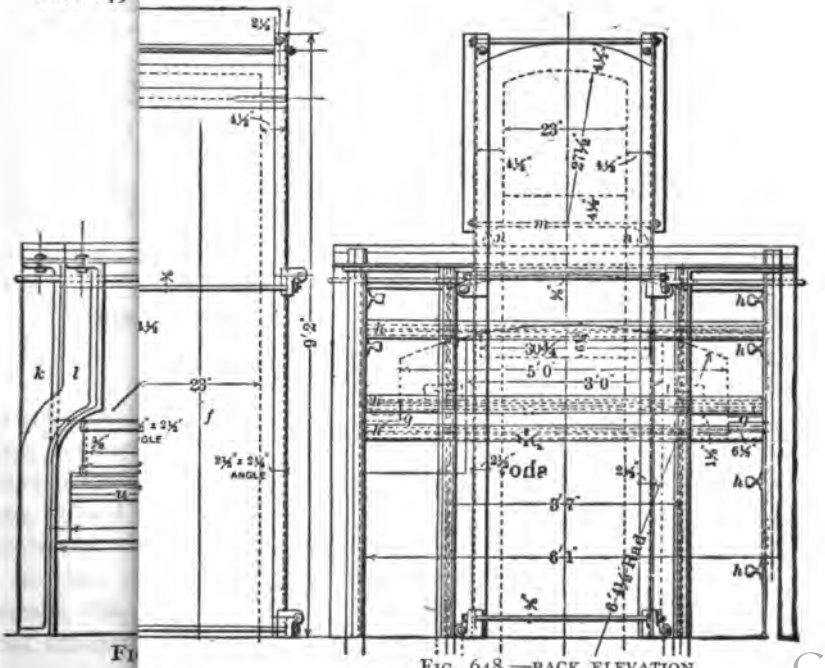

FIG. 648.—BACK ELEVATION

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A second furnace of the same type but oil-fired is given in Figs. 654–656; its water-cooled test and bed-plate resting on a Rhodes cupel-carriage are shown in Figs. 657–660; and the detail of the cast-iron bed-plate in Figs. 661–664. The construction of the furnace is much simplified owing to the absence of the fireplace. The hearth-opening, 6 ft. 2 in. wide by 8 ft. 3 in. long, has rounded corners at the back, and is circular at the front. At the back are two

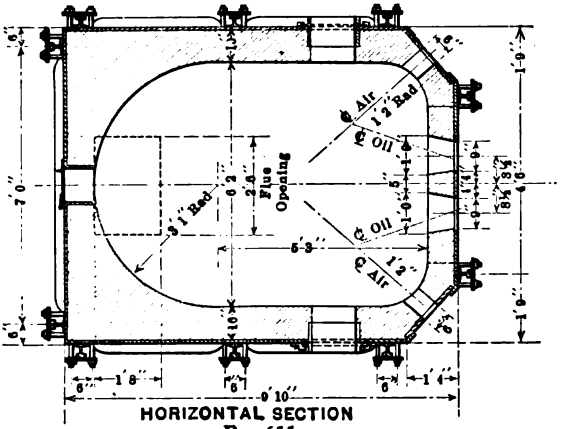

FIG. 655

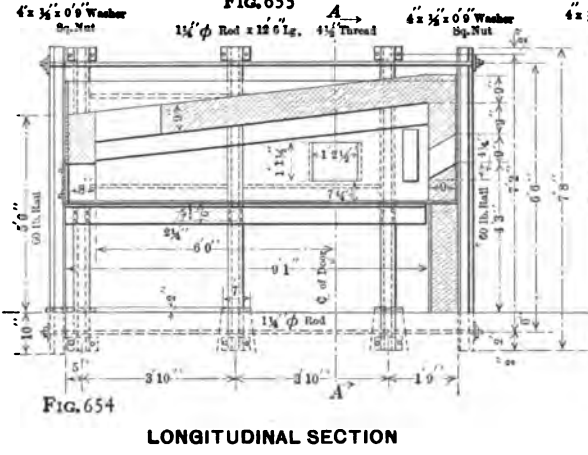

FIG. 654

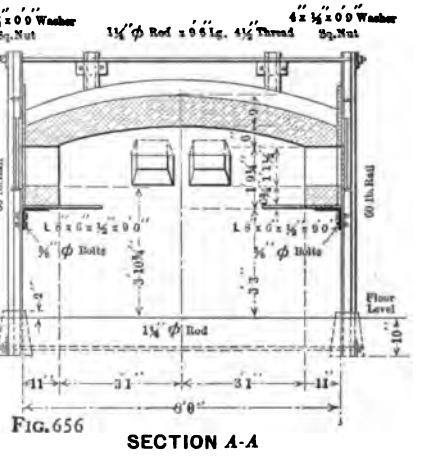

FIG. 656

FIGS. 654 to 656.—Modern English cupelling furnace, oil-fired.

ports for oil-burners, at the rounded corners two converging openings for blast-pipes; on each side is a working-door for access to the hearth and for pouring in retort bullion; at the front is the working-door. The side-walls rest on angle irons bolted to the buckstays tied top and bottom. The roof is supported by skewbacks similarly attached. It will be noted that in this furnace, as well as in that of the Pueblo S. & R. Co., Figs. 645–653, there is no brickwork beneath the side-walls, the space being left open. This is characteristic of most modern furnaces.

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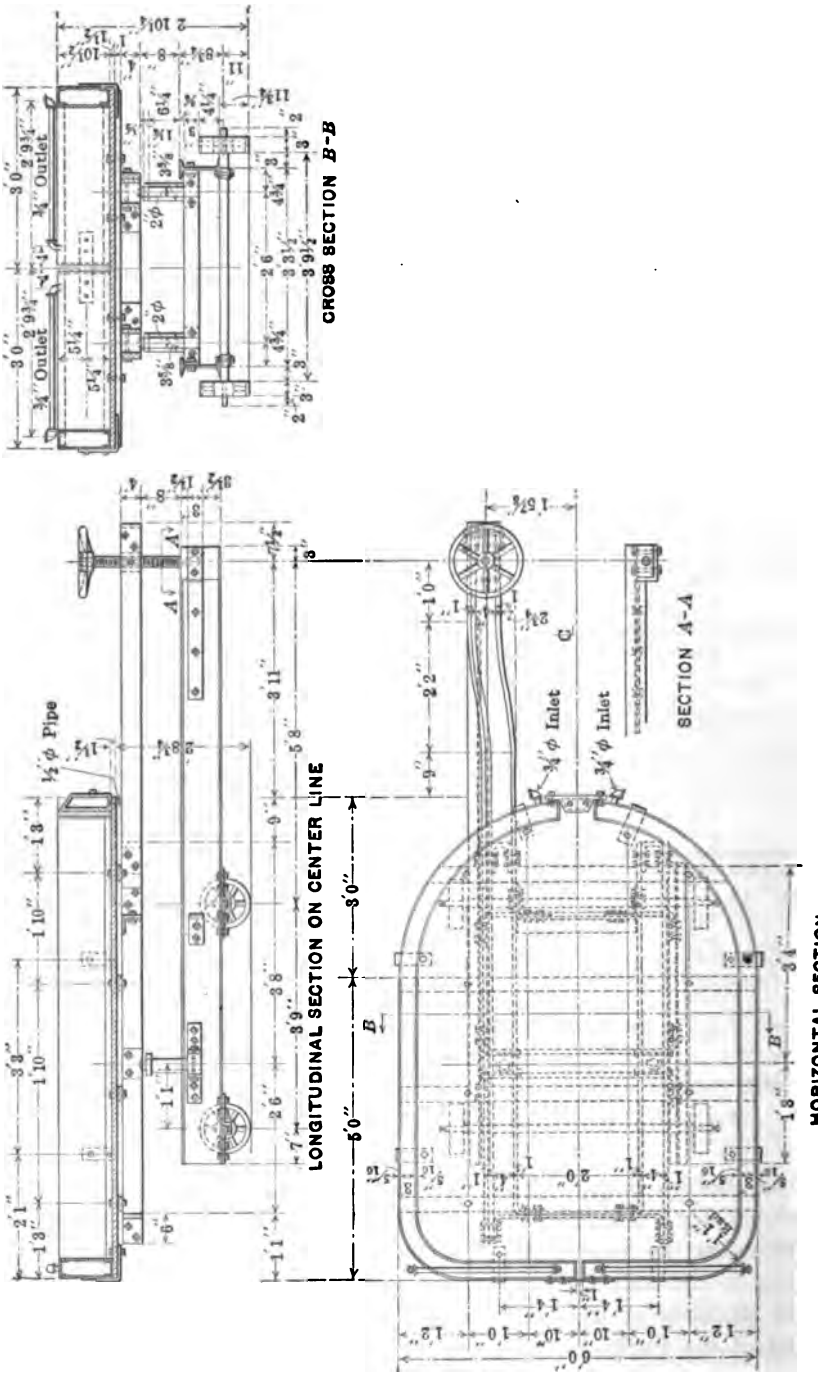

FIGS. 657 to 660.—Water-cooled test and bed plate on Rhodes cupel car of Figs. 654 to 656.

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Drawings of other furnaces, smaller in size and used in cyanide plants rather than in lead refineries, have been published by Weinberg,[1] Smith,[2] Munroe,[3] Clark, and Stanwood.[4]

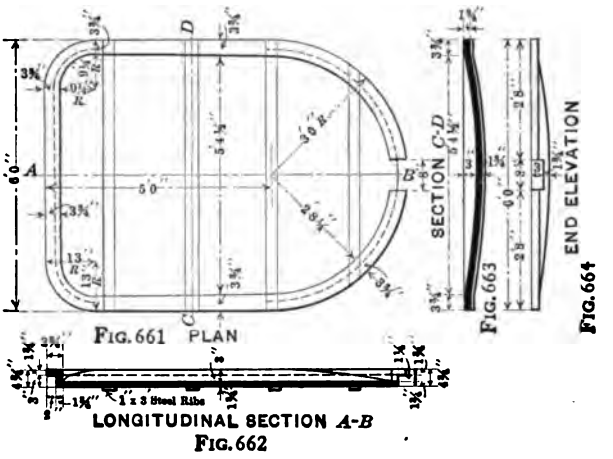

FIG. 661 FIG. 663 FIG. 664

FIG. 662

FIGS. 661 to 664.—Detail of cast-iron bed plate of Fig. 657 to 660.

280. Test Rings.—The test, which originally consisted of an oval frame of wrought-iron (4½ in. wide, ½ in. thick) filled with bone-ash, has undergone many changes in construction, manner of support, and filling material. In Figs. 665, 666, and 667 is shown a cast-iron test, *i*, resting on a Rhodes

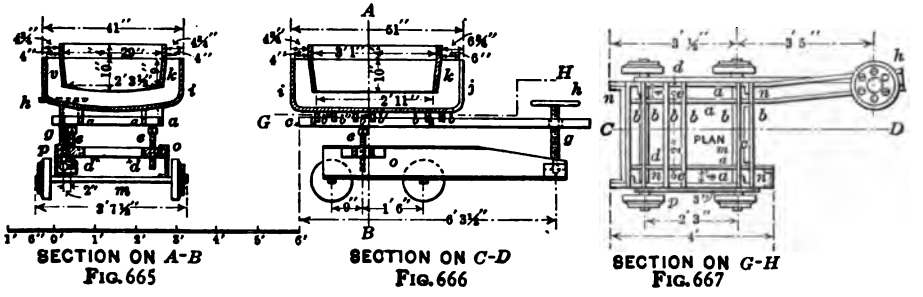

FIG. 665 FIG. 666 FIG. 667

FIGS. 665 to 667.—Cast-iron test on Rhodes carriage, National Works, South Chicago, Ill.

test carriage. It has a concave bottom and a cast-iron pattern of the cavity on the inside, both of which will be discussed further on. The test ring is rectangular in plan and has rounded corners. It fits into the cupelling furnace shown in Figs. 640–644. When in place its upper surface will be

[1] *Tr. Austral. Instit. Min. Eng.*, 1901, I, 167.
[2] *Eng. Min. J.*, 1913, XCV, 603.
[3] *Op. cit.*, 1137.
[4] *Tr. Inst. Min. Met.*, 1912–13, XXII, 68.

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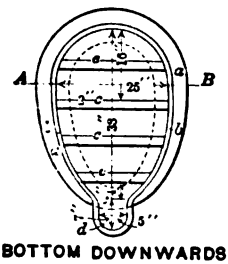
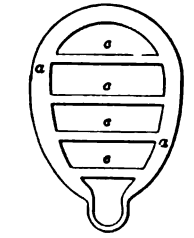
on a line with the upper rim of the compass ring, *p* (Fig. 641). In front it has a 3-in. slot, *j*, which is closed when the filling material forming the hearth is being rammed in. The rectangular form of test offers a large surface for oxidation, and therefore more lead can be cupelled on it than on an oval hearth.

Figs. 668–670 represent an oval cast-iron test ring, having a horizontal flange, *a*. When in place, the upper side of this will be close to or in contact with the lower side of the compass ring, *f* (Fig. 641), and the upper edge of the



BOTTOM DOWNWARDS
FIG. 668



BOTTOM UPWARDS
FIG. 669

FIG. 670

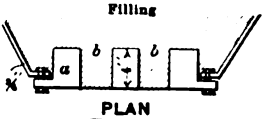


FIG. 671

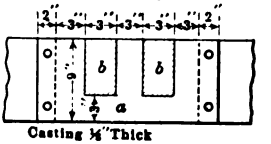


FIG. 672

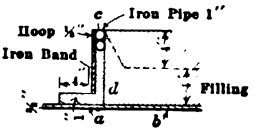


FIG. 673

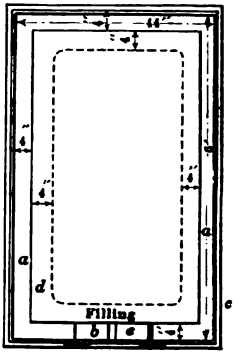


FIG. 674

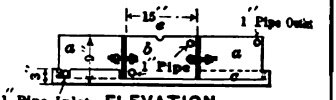


FIG. 675

FIGS. 668 to 670.—Oval cast-iron test, Pennsylvania Lead Co., Carnegie, Pa.
FIGS. 671 and 672.—Front of test, water-cooled in part.
FIG. 673.—Test with sides protected by water in cooling-pipes.
FIGS. 674 and 675.—Steitz water-jacketed test.

test ring itself will be in line with the upper edge of the compass ring, thus making the distance between the roof and the surface of the lead as small as possible. The test ring protrudes over the horizontal flange at *c*, forming a loop, *d*. In tamping in the filling material the loop is not filled, in order that the litharge overflowing from the hearth may pass through it into the litharge-pot below; thus any contact between hot litharge and iron frame is avoided. Across the bottom of the test ring are four cast-iron arms, *e*, to hold the filling in place.

With both tests the corrosive action of the litharge very soon eats out the filling, especially at the front, and in a comparatively short time the test has

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to be removed from the furnace and replaced by another. To counteract the corrosion of the front, and at the same time permit the raising or lowering of the litharge gutter, a water-jacket, as shown in Figs. 671 and 672, is fastened by means of bolts to the test ring, be this cast iron or wrought iron. The jacket has the same depth as the test ring and forms the breast. The litharge runs off through gutters cut into the filling, *b* (Fig. 672). As this is cooled by the water circulating in the jacket, it is eaten out only very slowly. The jacket itself does not come in contact with the hot litharge in the furnace, as it is protected from it by a 3-in. rim of filling. This wears out somewhat, but never or rarely so far as to bring the casting into direct contact with the hot litharge.

By this arrangement only the front is protected. A device that protects the sides alone is shown in Fig. 673. Here *a* is a cast-iron test ring, resting on a bed-plate, *b*. It is surrounded by an iron hoop tied by an iron band. On top of the test ring are placed two 1-in. pipes in which water circulates. The filling, *d*, is rammed in the usual way. The lead, while the furnace is running, being always kept about at the same height, the litharge can show its bad effect only on the level of the water-pipes, and these counteract to a great extent the corrosive action.

A combination of the two arrangements is shown in Fig. 647. Going one step further in water-cooling we come to the Steitz water-jacket test, as represented in Figs. 674 and 675. Here *a* is a rectangular water-jacket made of boiler iron. The open space in front is closed by a cast-iron or bronze water-jacket having a litharge gutter, *e*; it is fastened with bolts to the wrought-iron jacket. The jackets are placed on a cast-iron test plate, *c*, which supports the filling, *d*. Both jackets are protected from the hot litharge forming on the surface of the lead in the furnace. The gutter, *e*, alone is attacked by the litharge, and is eaten out after some time; the breast jacket then has to be exchanged for another. This can be done quickly while the furnace is in operation. The effect of jacketing is that the filling lasts longer than with the other test rings. The cast-iron front jacket, in addition to preserving the breast better than any of the other arrangements, has, as regards concentration of lead bullion, another advantage in that the depth of the litharge gutter does not have to be regulated by the cupeller, but always remains the same. This renders it, however, unavailable for bringing very rich, say 70-per cent., bullion up to fine silver, as the uniform level of the gutter prolongs indefinitely the removal of the last lead contained in the silver.

A rectangular steel jacket which has to expand and contract is likely to develop cracks or leaks at the corners. In more recent jackets corners are therefore avoided. A modern steel jacket, which fits into the open hearth-space of the furnace, has been shown in Figs. 657–660, and its bed-plate in Figs. 661–664. The jacket is supporteb by a Rhodes carriage (§281). The jacket is 8 ft. long, 6 ft. wide, and 10½ in. high; it is made of $\frac{5}{16}$-in. steel and has a water-space 3 in. wide served by a ¾-in. pipe. It consists of two parts joined at the back and connected at the front by a cast-iron water-jacket.

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281. Test-ring Supports.—The manner of bringing the tests into position and fixing them there has undergone various changes. The old method consisted in driving four iron wedges between the bottom of the test ring and two transverse bars, the ends of which were inserted 4 in. below the test frame into the firebridge wall, *d*, and the flue wall, *e* (Fig. 641). It is not much used now.

Sometimes wedges are still retained to adjust the large test frame represented in Figs. 674 and 675. This is placed on two 9-in. brick walls running along the firebridge and flue walls. It is then raised to the height of 12 in., and four pillars, each three bricks high, are placed beneath the bed-plate, which brings it up nearly to the compass ring. By then driving flat wedges between the bed-plate and the brick posts the test is adjusted to its final position. A more common way with small tests is to use jackscrews, which are satisfactory if the entire hearth is made accessible as in the modern cupelling furnace where the hearth walls rest on cast-iron supports. The screws, 18 in. long and 1½

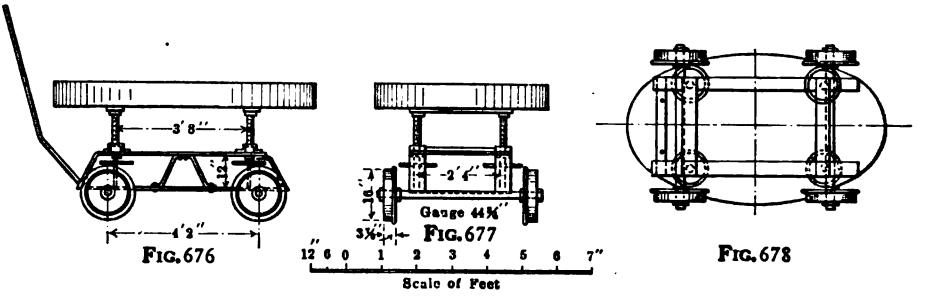

FIG. 676 FIG. 677 FIG. 678

FIGS. 676 to 678.—Test-carriage of Frazer & Chalmers.

in. in diameter, work in two transverse bars placed 12 in. beneath the test ring. They are similar to scews, *d*, in Fig. 679. The test ring rests on a cast-iron plate into which the points of the four screws are set. Four screws are also found in connection with a test carriage, as shown in Figs. 676–678. Here the test is easily brought into position, and then raised by means of the screws and brought up close against the compass ring.

With the three arrangements described, the test, when once placed, is immovably fixed while the cupellation is proceeding. Further, when the furnace has become hot, the turning of the four screws sometimes presents difficulties, which have been largely overcome by more perfect air-cooling.

Replacing the two front screws by a suspension or supporting device which can be raised or lowered has overcome the fixed character of the older tests.

The Lynch[1] test support is represented in Figs. 679 and 680. The test plate, *e*, shown bottom side up in Fig. 680, has at the back the two sockets for the points of the two back screws, *d* (Fig. 679). Two support bars, *c*, diverging 27 in. when extending in front of the furnace, are clamped to the front half of

[1] Blake, *Tr. A. I. M. E.*, 1881–82, x, p. 220; U. S. Patent No. 275232, April 3, 1883.

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the plate. When in place they are hooked with the turn-buckles, *b*, to a triangle made of ¾-in. iron, which is attached to a differential pulley, *a*, hung from the roof. With the pulley the test can be raised or lowered quickly and evenly to regulate the flow of litharge without altering the depth of the litharge gutter. The contents of the test (rich lead or silver) can also be poured. With the turn-bolts the test can be tilted sideways to counteract the action of the litharge, should this corrode one side of the hearth more than the other.

The suspension support of the Pueblo cupelling furnace has been shown in Figs. 645–653.

Another movable test support, which has been adopted by most refineries, is the one by F. B. F. Rhodes, represented in Figs. 665–667. The test is supported by a carriage, *o*, with its movable upper frame, *aa'*. This rests at the

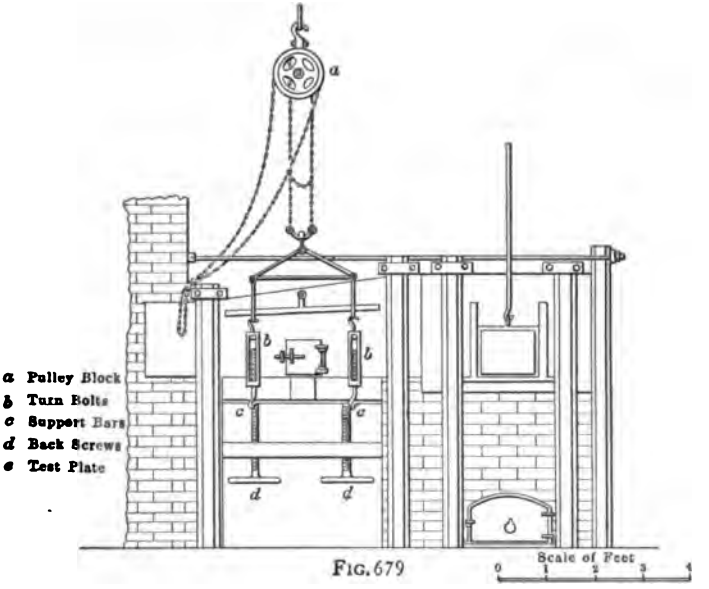

FIG. 679

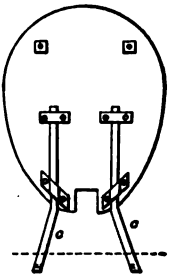

FIG. 680

FIGS. 679 and 680.—Lynch test-support.

back on two screws, *e* and *e'*, working in the blocks, *d*, and, *d'*. At the front, it is supported by the screw, *g*, working in the right arm, *a*, of the frame, which extends a short distance in front of the furnace, and is turned outward sufficiently for the wheel, *h*, to be to the right of the cupeller. By inserting a hook into one of its circular openings he turns the wheel to right or left, and thus raises or lowers the front of the frame, *aa'*, and with it that of the test. Into the upper frame cross-beams, *b b' b''*, are let in, which serve as support for the concave bottom of the test. If this were straight, the surface of beam, *a*, would represent in section an unbroken line, two more beams like *c* and *c'* being sufficient to give the frame the required stability. The wheels of the carriage are grooved and travel on two rails running across the space, *c*

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(Fig. 640); thus all shifting of carriage to right or left, when placed in correct position, is avoided.

The support at the back by screws, *e*, has the disadvantage that when the frame has to be lowered there before the carriage is taken out, much difficulty is experienced in turning the screws. To avoid this, the two screws, *e*, have been replaced by two pivots which fit into circular openings of two blocks fastened to the beams as shown in Figs. 654–656. When the frame, resting at the back on these two pivots and supporting the test, has been wheeled into its correct position and is to be raised, this is done by lifting it a few inches with a crowbar or jackscrew and placing two U-shaped castings of the correct height, around the pivots. On withdrawing or lowering the crowbar, the frame sinks on the castings, which then support it. When the test is to be exchanged, the frame is again slightly raised, the two U-shaped castings are removed, and the frame is slowly lowered, when it will again be supported by the pivots. The test ring is then sufficiently low for the carriage to be withdrawn from under the compass ring.

Another movable support with specially constructed test ring has been constructed by Roesing,[1] and is used at Tarnowitz, Silesia.

Movable tests are in much favor when the test ring is not water-cooled, *i.e.*, where the level of the lead is gradually lowered by cutting deeper the litharge gutter; with test rings that are entirely water jacketed they have not been used until lately.

Whatever test support may be in use, care should always be taken to plaster over with clay the upper surface of the ring, so as to protect it against coming in direct contact with the flame, and at the same time to prevent the air from rushing in between compass ring and test ring.

282. Hearth Material.—The requirements that the hearth material should fulfill have been given on page 596. The leading materials are the following:

1. *Bone-ash.*—This material was used originally in the English cupelling furnace. It is still used in some English plants, but has been given up in most refineries on account of its cost and the fact that it is not well suited for large tests. The bone-ash is ground to pass a 26-mesh screen, moistened with water which may or not contain pearl ash amounting to 2¼ per cent. of the prepared bone-ash. The moistened material is rammed into the test-frame and the cavity scooped out in the same manner as with the limestone-clay mixture detailed below.

2. *Limestone-clay.*—A mixture of 3 vol. of limestone and 1–2 vol. of clay, depending upon the binding power of the latter, is ground through a 12-mesh screen.

In filling the test-ring, the mixture is moistened, as shown in §277, and then tamped into the ring. Some works beat down the mass in three separate layers; others add the necessary material all at once and begin then with the tamping. If the cast-iron test ring (Figs. 668–670) is to be filled, a piece of wood having

[1] *Berg. Hüttenm. Z.*, 1883, XLII, 577; *Eng. Min. J.*, 1884, XXXVII, 41.

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the form of the loop, *d*, is put in place, and then the filling beaten down. When finished, the block is taken out, leaving open the slot for the discharge of the litharge. The wood is best withdrawn only when the test has somewhat dried, as then there is less danger of breaking off part of the filling. The tamping-irons are about 4 in. in diameter at the base. The test is filled entirely with the material, and the cavity then scooped out with a trowel. A very good way to insure a uniformly hard surface is to place a wooden frame on top of the test ring, and then partly fill it with the hearth material. When the frame has been removed, the excess of material is scraped off down to the test ring, and the cavity then scooped out. A rim from 3 to 4 in. wide at back and sides, sometimes 6 and even 10 in. at the front, is left untouched. The cavity has its lowest point near the front to facilitate the dipping out of concentrated bullion or fine silver. The filling material should be at least 4 in. thick at the lowest point, and the depth of the cavity about 5 in. Thus an oval test 4 ft. 6 in. long, 2 ft. 1 in. wide, and 5 in. deep, holds about 2,500 lb. of lead.

3. *Portland Cement.*—The cement best suited is one that is slow-setting. At some works a mixture of cement and ground brick is used in order to give the lining more elasticity. The gain in mechanical strength is made at the expense of resistance to corrosion, as the SiO_2 of the brick renders the mixture less basic.

In filling the test with cement, or with a mixture of cement and ground brick, this is moistened and tamped down in the usual way, the cavity, however, being formed during the tamping. A quicker and better method is to place the test bottom upward over a mold having the form of the cavity, and then to beat down the moistened cement. In using cement, it is essential that the work to be done quickly, as it must be finished before the cement shows any signs of setting.

At other works the mixture used for the bottom is 1 vol. cement, 2 sand, and 2 broken, chilled, glassy, blast furnace slag (pieces ¼ in. thick, ½ in. square). The base-plate being concave, the bottom is 11 in. thick along the center line from back to front. The sides, 12 in. thick, are made up of a mixture of 10 barrows 6-mesh limestone and 2½ sacks of cement; the breast material is the usual limestone-clay mixture. The test, 8 by 10 ft. and 17 in. deep, holds 3 tons of lead.

At a third plant the bottom is rammed with a mixture 12 vol. crushed limestone, 6 fire-clay, and 4 cement; on top of this comes a layer of brick composed of compressed bottom-mixture followed by a layer of magnesite brick; the sides are rammed with bottom-mixture. The life of the lining is from 60 to 70 days provided it is patched.

At a fourth plant the bottom is lined with fire-brick laid dry and grouted thoroughly with fire-clay. The sides are of cement rammed around a cast-iron frame as shown in Figs. 665 and 666.

4. *Magnesite Brick.*—At one plant, bottom and sides are of magnesite brick. This material resists corrosion better than any of the mixtures given. It is, however, expensive and necessitates a test made of a single casting, Figs. 665

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and 666, as it is difficult to lay the bricks so closely that no lead will pass out of the hearth through the joints.

When a test is filled, it has to stand for a fortnight and longer in a warm place (usually the cupelling-room) to dry. This is essential, as the filling is likely to crack if used too soon after it has been put in. Before it is to be used in a cold furnace, a small charcoal fire is made on it. In a warm furnace the fire is kept low for 3 or 4 hr. after the test is in place.

283. The Blast.—The blast was originally produced by a steam jet. This has given way entirely to a blower. The machines in common use are the Baker and Root blowers, and the Sturtevant fan; blower and engine are usually supported on the same bed-plate. The pressure of the blast is about 4 oz. per sq. in. The blast-pipe is of sheet-iron and about 3 in. in diameter. It is frequently fitted into a cast-iron nozzle, sometimes water-cooled, which delivers the air through an aperture 4 in. long and ½ in. wide. This is better than simply flattening the end of the sheet-iron pipe.

284. The Tools.—The tools required by the cupeller are few; two rods (7 ft. long, of ½-in. iron), bent to a hook; one chisel-pointed bar (7 ft. long, of ¾-in. iron); one saw (9 in. long, ½ in. wide, 2 in. deep), attached to a handle (6 ft. long, of ½-in. iron), to cut down the breast; one fire-hook (10 ft. long, of 1-in. iron, with a 4- by 10-in. head); one scoop; one ladle; 10 bullion molds or silver molds, and three or more litharge buggies having small kettles (13 in. in diameter and 8 in. deep) to hold the litharge.

285. Mode of Operating.—When the test is in place and well warmed, the temperature of the furnace is gradually brought to a dark-red, and some lead introduced through the front and melted down. When this has become a cherry-red, the blast is put on, and cupellation started. No distinction is made between dross, skimmings, and litharge, as in the German cupellation. The litharge is made to run off at the front, and fresh lead is supplied from the back, where one or two small bars protruding through openings into the furnace are melted down at such a rate as to keep the lead in the test always on the same level. The litharge is collected in a cast-iron pot running on wheels. With the large furnaces in common use today an ordinary slag-pot with a loose central partition wall has replaced the small litharge-pots. The object of the partition is to facilitate the breaking of the cone of litharge. The litharge-pot was at one time replaced by a water-box. This has two advantages. It reduces the temperature for the cupeller and presents the litharge in a granulated form, which is easily handled and sampled. It has, however, the disadvantage that a cupel-carriage cannot be used, and that in the blast furnace there is more loss in lead and silver when granular litharge is charged than if it is in lump form. Granulating has been abandoned.

With a stationary iron test ring, the litharge is run off through a gutter cut into the lining. It is not often that one gutter alone serves for the passage of the litharge; generally there are three and often four, opened one after the other to prevent excessive corrosion. A movable test gives an additional mode of regulating the flow of litharge by lowering and raising the front. With the

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Steitz water-jacket test the flow of the litharge is regulated only by the quantity of the lead that is melted off from the bars at the back of the furnace. The gutter can be closed for a short time by allowing litharge to accumulate there, or with a piece of clay.

The flow of the litharge is so regulated that about one-half the surface of the lead remains covered. The former practice of cupelling and fining in the same furnace has been abandoned at all large works. At present it is common to concentrate the bullion to 60 or 70 per cent. of Ag on one test and to fine it in a separate furnace. For this concentrating work, the water-jacket test is excellent, as it can be run by an inexperienced man, whereas judgment and practice are necessary with a test where the litharge gutter has to be regulated by the cupeller. By thus dividing the cupelling into the two operations of concentrating and finishing, a smaller number of experienced and reliable cupellers is necessary. When the bullion is concentrated to the desired degree, it is ladled or poured out and goes to the finishing furnace, and the concentration furnace is again filled. Thus a concentrating furnace runs constantly. After a certain time the bottom becomes too thin and has to be exchanged. A test ring filled with limestone clay, if used 4 or 5 hr. daily for finishing, lasts only 30 days; a cement-test used for the same purpose lasts months. A water-jacket test filled with limestone clay, used for concentrating, lasts about 60 days. A test ring filled with cement and used for concentrating and refining lasts 7 days.

The finishing is always done on a test having an iron test ring. The operation is the same as in concentrating, but usually not continuous. Toward the end, when the silver-lead alloy becomes less readily fusible, the temperature has to be considerably raised. When the silver has become sufficiently concentrated, the addition of rich bullion is stopped. The last litharges are drawn off and the test remains almost filled with crude silver, which has now to be fined. It is not often that the brightening is seen. Samples taken from the metal bath show how far the cupellation has progressed. The fining consists usually in exposing the silver for some time to the action of the heat and the blast. Bone-ash is sometimes given in small quantities to absorb the impurities that float on the surface or have collected on the edge. At some works nitrate of soda is used in the furnace to make the silver at least 997 fine, as this has become the standard below which fine silver should not go. The niter is spread on the silver, a shovelful (about 12 lb.) at a time. To prevent the soda from corroding the filling of the test ring, some refiners spread finely ground brick over the niter. The slag, floating on the silver, is removed only when the silver is fine and ready to be cast into molds. About 15 shovelfuls of niter are required for 50,000 oz. of silver.

In this connection the behavior of tellurium is of interest. Whitehead[1] and Ulke[2] state that in the second refining of doré silver (obtained in working the anode mud from an electrolytic copper refining plant) with niter, the resulting slag contained as much as 20 per cent. Te in the form of tellurite of sodium.

[1] *J. Am. Chem. Soc.*, 1895, XVII, 849.

[2] *Eng. Min. J.*, 1896, LXII, 512.

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It is very difficult to remove all the tellurium from the silver, owing probably to the presence of Cu. Niter alone does not appear to meet the need; refiners are often forced to blow air through the silver for hours at a time to get rid of all the Te.

A more simple procedure, which is effective and in common use in European plants as well as in some works in the United States, is to add some fresh lead to the doré silver, when this is about 990 thousandths fine. A charge of 60,000 oz. doré silver requires about 1000 lb. of lead, which is added in three or four portions. The litharge formed oxidizes the Cu, and with it the Te and Bi which adhere so strongly to the silver.

Though it is difficult to remove Te, the metal Se is readily oxidized and volatilized, and is found in the flue-dust. The dust often forms the raw material for the production of selenium.[1]

As regards copper, the investigations of Mathewson and Stokesbury[2] are of interest. They find that Cu_2O is readily soluble in Ag when molten, and practically insoluble when solid. The freezing-point curve shows a eutectic point at 945° C.; the eutectic is composed of Ag 98.7 and Cu_2O 1.3 per cent. A comparison of calculated values with those obtained by experiments indicates that dissolved Cu_2O is either dissociated or reacts upon Ag as shown by $Cu_2O + 2Ag \rightleftarrows 2Cu + Ag_2O$. A Ag–Cu alloy with little Cu, melted and allowed to solidify in air, shows that the Cu occurs as Cu_2O between grains of Ag; hence, the Cu in all cupelled silver must be present as Cu_2O and not as Cu as has been supposed.

The indications of fine silver are: that the surface is smooth and clean; that stirring fails to bring impurities to the surface; that a tool held over the silver is clearly reflected in it; that a sample taken by inserting a rod will show no spots whatever on the surface, and have a pure, silver-white color; and that a sample taken with a spoon will spurt while cooling, although this is not a good test. Some refiners cast a small sample-bar, examine the surface, which should be smooth, and the fracture, which should be finely granular and show a silky luster; test for malleability by hammering, etc. The only way to know definitely how the fining is progressing is to make an assay. This is done in the dry way, by weighing out twice ½ gram of a granulated sample and ½ gram of c. p. silver as a check, and cupelling the three samples with the same amount of lead on three cupels placed in a row in the muffle. A second assay half an hour later will show whether any progress has been made. An assay in the wet way, with potassium-sulphocyanide, using ferric sulphate as an indicator, will give the same result more quickly than cupelling.

An analysis[3] of government silver (assay silver) showed: Ag 99.929, Cu 0.056, Pb 0.003, Au 0.007, As 0.001, Sb 0.002, Fe 0.001, Zn trace, Te 0.001 per cent.

When the silver is fine, that is, when it ranges between 997 and 999½, it

[1] See Hofman, "Copper," 1918, 518.

[2] *Internat. Zt. Metallogr.*, 1914, v, 193.

[3] *Brass World*, 1906, II, 389.

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is either ladled out into warmed molds, or, if the Lynch test-support is used, it can be poured. Sometimes this is done into water, to be remelted at a lower temperature in a plumbago crucible, a new retort (heated in a tilting furnace), or a separate cupelling furnace. If in the last, the granulated silver is covered with charcoal. If the silver is to be poured or ladled from the test in which it has been refined, it is advisable, if smooth bars are to be obtained, to cover it about 20 minutes before pouring with charcoal. While filling the molds, which have been warmed and smoked, a small stick is held in contact with the surface of the silver to collect floating slag or impurities. When a mold is filled, a small piece of paraffine is thrown on the metal and the lid quickly put on.

The sample of the fine silver is best taken from the mold. When this has been filled, a long-handled iron spoon is inserted, the silver stirred with it, and the sample taken out and poured on a wet board resting in an inclined position on the rim of a basin filled with water (Eurich). The granules collecting in the basin are very small and make weighing of a definite quantity for the assay a quick and easy operation. Taking chips from different parts of the bar is unsatisfactory, as the impurities are as a rule not evenly distributed.

286. Intermediary Products.—The intermediary products of the cupellation process are litharge, cupel-bottom, and flue-dust. Litharge coming from retort bullion runs, when it is pure, from 50 to 60 oz. Ag per ton; when it is impure, *e.g.*, when drosses of the retort bullion are being scorified, often from 150 to 200 oz. The cupel bottom varies too much in Pb and Ag to give any average figure. Samples of fume and dust from cupelling rich retort bullion gave the values shown in Table 150.

TABLE 150.—ASSAYS OF DUST AND FUME FROM CUPELLING

	Pb per cent.	Ag, oz. per ton	Au, oz. per ton	Authority
Dust	21.6	20.0	0.16	F. B. F. Rhodes
Dust	27.6	106.6	0.40	M. W. Iles
Gray fume	41.8	134.3	0.32	M. W. Iles
Red fume on cupel door	59.0	1646.0	.....	M. W. Iles

These products almost always go to the ore blast furnace; litharge is sometimes used, as already stated, to hasten the softening of lead bullion that is especially hard.

287. Results.—In the test 4 ft. 6 in. by 3 ft. 6 in., Fig. 643, 7000 lb. of retort bullion are cupelled by three men in 24 hours, using from 1½ to 2 tons of coal, according to the quality of the fuel. On a larger test, 7 ft. by 4 ft. 10 in. and 4 in. deep, holding from 1500 to 2000 lb. retort bullion, 6 tons are cupelled in 24 hours with 1½ tons of coal. In a similar test, 7 ft. by 5 ft. 4 in., 6½ tons 2000-oz. retort bullion are cupelled in 24 hours, 39 gal. of reduced oil, atomized by air, being consumed for every 1000 oz. silver produced. In the test 6 by 8 ft.,

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shown in Figs. 657–660, which holds 5 tons lead bullion, there are cupelled in 24 hr. from 5 to 6 tons with an oil consumption of 280 gal.

It is advisable to have 8-hr. shifts for cupellers, to prevent their becoming leaded.

The concentrating of 1000 lb. of 70-per cent. bullion on a 33- by 28-in. test, 5 in. deep, and refining of the resulting silver (say 12,000 oz. silver), lasts about 5 hr., requires one man, and about 1500 lb. of nut coal. When the finishing-furnace is stopped for a day or two, the fire on the grate is kept going in order that the temperature of the test may not sink below a dull-red; charcoal is often kept aglow on the test, as it makes it again porous when it is much soaked with litharge.

288. Loss.—The loss of Pb in cupelling is generally given as being 5 per cent.; that of Ag is under 1 per cent., presupposing that the uncorrected assay to be the basis of calculation; the loss in Au is nil. Rose[1] found that Au began to become volatile just below 1100° C., that the loss amounted to 0.02 per cent. at 1200°, and that standard copper-gold was more volatile than pure Au. Mostowitsch and Pletneff,[2] on the other hand, found that no Au was volatilized at these temperatures.

289. Comparison of German and English Cupellation.—A comparison of the two methods of cupelling leads to the conclusion that the German method is by far the more expensive for purposes for which cupelling is generally used today. Although it forms litharge more rapidly, because the hearth is larger, it produces only a comparatively small amount of silver as the product of one operation. To remove the silver, the furnace has to be cooled, and the hearth torn out and replaced by a new one. This takes time, thus neutralizing the advantage of the quick formation of litharge, and costs much labor, fuel, and material, including a large amount of hearth material, which has to be smelted in the blast furnace for every cupellation. In the English cupelling furnace, especially with its American modifications, a cupel bottom lasts for weeks; the process is therefore less interrupted, and thus much expense for labor, fuel, and material saved, but it has the drawback that the litharge is always more apt to be rich and impure.

A German cupelling furnace might be in place when the resulting litharge is to be sold as such, and is therefore required to be pure, and poor in Ag, but modern American cupelling furnaces are often as large as the older German, hence this consideration has lost its former importance. The English furnace with American improvements is decidedly preferable, if the bullion to be cupelled is so rich that the resulting litharge, in any case, would run too high in Ag to be sold as such. In this case, and it is the common one to-day, it is not of much consequence whether the litharge be a little poorer or richer in Ag, or if it be somewhat contaminated with impurities, as long as the advantages more than make up for such deficiencies.

[1] *J. Chem. Soc.*, 1893, LXIII, 714.
[2] *Met. Chem. Eng.*, 1917, XVI, 153.

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BETTS PROCESS

290. Betts Process, General.[1]—This process is the only electrolytic process of refining lead bullion which has been successful. It is in operation at Trail, B. C.; East Chicago, Ind.; Omaha, Neb., and New Castle-on-Tyne, England. Other processes have either been failures, or have not gone beyond the laboratory stage.

The process is worked according to the multiple system common in the electrolytic refining of copper.[2] Cast anodes of lead bullion and lead cathodes (of electrolytic lead), cast in sheet form, connected in multiple, are suspended from copper bars across an oblong vat, which is charged with an electrolyte containing lead fluosilicate ($PbSiF_6$) and free hydrofluosilic acid (H_2SiF_6). The current enters the anodes, passes through the electrolyte to the cathodes, dissolves the lead from the anodes, and deposits it on the cathodes. The impurities, which nearly all adhere to the undecomposed anodes, form the anode mud or slime. The cathode lead with the starting sheet is melted and cast into bars; the anode mud, recovered from the non-corroded anode and collected from the bottom of the tank, is refined to recover precious metals, Sb, As, Bi, Cu, and possibly Se; the Te, for which there is no market at present, goes to waste.

In the process the $PbSiF_6$ dissolved in water acidulated with H_2SiF_6, is in part dissociated into $Pb^{\cdot\cdot}$ and SiF_6'' ions. In the electrolysis SiF_6'' ions travel to the anode, give up their charges to the anode, and combine with an equivalent amount of Pb. In the same manner the $Pb^{\cdot\cdot}$ ions migrate toward the cathode, give up their charges to the cathode, and are deposited as metallic lead. In reality the process is not as simple as outlined; there occur side-reactions causing a deposition of SiO_2 on the anode, which still lacks a satisfactory explanation.

Electrolysis causes Pb, Sn, Zn, Fe, Ni, and Co to go into solution, while Cu,

[1] Betts, A. G., "Lead Refining by Electrolysis," Wiley, New York, 1908, General.
Betts, U. S. Patents, Nos. 713277 and 713278, Oct. 9, 1902.
Ulke, *Eng. Min. J.*, 1902, LXXIV, 475.
Betts, *Tr. A. I. M. E.*, 1904, XXXIV, 175.
Senn, *Zt. Electrochem.*, 1905, XI, 229; *Min. Mag.*, 1905, XII, 71; *Electrochem. Metal. Ind.*, 1905, III, 272.
Whitehead, *Mines and Minerals*, 1905, XXV, 285.
Betts, *Electrochem. Met. Ind.*, 1905, III, 441.
Wolf, *West. Chem. Met.*, 1907, III, 83.
Mardus, G., "Doctorate Thesis," Berlin, 1908, using HBF_4.
Betts, U. S. Patents, Nos. 891395 and 891396, June 23, 1908; 918647, April 20, 1909, Treatment of Slime.
Betts, *Metallurgie*, 1909, VI, 233.
McNab, U. S. Patent, No. 905753, Dec. 1, 1908, Recovery of Sb from slime.
Foerster-Schabe, *Zt. Electrochem.*, 1910, XVI, 279, Treatment of Bi-Ag.
Kern, *Met. Chem. Eng.*, 1911, IX, 417, Anode Mud.
Miller, *Min. Eng. World*, 1913, XXIX, 57.
Thum, U. S. Patent No. 1098854, June 2, 1914, Treatment of Cu-bearing Bi.
Smith, U. S. Patent No. 1166721, Jan. 4, 1916, Refining of Bi.
Ruff-Braun, *Ber. deutsch. chem. Gesellsch.*, 1914, CCXLVII[2], 646, Preparation of HF.

[2] See Hofman, "Copper," 1918, 491.

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Sb, As, Bi, Cd, Ag, Au, Se and Te remain with the anode. Of the metals that are dissolved, Sn stands so near Pb in the electrolytic series that it is deposited with the Pb. According to Betts,[1] the small amount of 0.2 per cent. Sn in the anodes of Trail, B. C., went over to the cathodes; hence Sn-bearing lead bullion has to be softened (§241) before it is cast into anodes. The other metals, Zn, Fe, Ni, and Co stand well above Pb in the series and are therefore not precipitated from the solution with Pb. Of the metals which remain insoluble, small amounts of Sb are likely to be carried over mechanically to the cathode especially with a current of high density (17 to 18 amp. per sq. ft.) and a warm electrolyte (37 to 38° C.),[2] so that the cathode lead may have to be refined for a short time, *e.g.*, poling with air in a kettle (§252), in order to remove them; any Sn present would be expelled from the Pb before Sb.

291. Electrolyte.—The electrolyte contains from 7 to 10 per cent. Pb and from 8 to 12 per cent. total fluosilic acid (H_2SiF_6); the free H_2SiF_6 varies from 3 to 5 per cent. A convenient factor to approximate the H_2SiF_6 in combination with Pb is to multiply Pb by 0.7. Curves showing the electric conductivities of $PbSiF_6$ and H_2SiF_6 have been given in Figs. 30 and 31. To the electrolyte gelatine is added as a hot strong solution of glue to the amount of 0.1 per cent. in order to obtain a solid deposit.[3] The daily addition in practice is about 0.013 per cent. of the weight of the electrolyte. The electrolyte is made at the works by allowing HF to act upon SiO_2 to form H_2SiF_6, and this to combine with Pb or PbO or white lead ($xPb(OH)_2+yPbCO_3$) to produce $PbSiF_6$. The hydrofluoric acid is either the commercial product with 33 per cent. HF, or, it is made at the plant by allowing H_2SO_4 to act upon fluorspar (CaF_2). The reaction taking place is expressed by $CaF_2+H_2SO_4=2HF+CaSO_4$. According to Ruff and Braun[4] the process taking place is not as simple as is generally held. He found that the best yield, 81 per cent. of the theoretical, was obtained by using 90 per cent. of the amount of the H_2SO_4 (sp. gr. 1.824) called for theoretically and heating for 3 hr. to 200° C. A horizontal cast-iron cylinder 10 by 4 ft. fired from the bottom is charged with CaF_2 and H_2SO_4. As the reaction develops much heat, the fire has to be well regulated. The liberated HF is collected in two leaden Wolf flasks connected in series. In the first flask is found some H_2SiF_6 which comes from the SiO_2 of the CaF_2; in the second, which is water-cooled, the HF is condensed.

The HF is made to ascend in a small lead tower filled with pure sand (SiO_2, 99.5 per cent.) at such a rate that it dissolves the SiO_2; the overflowing H_2SiF_6 is conducted into an oblong box filled with granulated lead where there is formed $PbSiF_6$; the solution is circulated by means of a submerged centrifugal pump

[1] Treatise, page 47.

[2] Some refiners hold that the transfer of Sb to the cathode is not wholly mechanical.

[3] Senn, *Zt. Electrochem.*, 1905, XI, 229; *Min. Mag.*, 1905, XII, 71; *Electrochem. Met. Ind.*, 1905, III, 272.

Mathers and Overman, *Trans. Am. Electrochem. Soc.*, 1913, XXIII 153; *Met. Chem. Eng.*, 1913, XI, 285.

Mutscheller, *op. cit.*, 1915, XIII, 353.

[4] *Ber. deutsch. chem. Gesellsch.*, 1914, CC, XLVII², 646.

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made of copper and provided with a bronze shaft. The electrolyte is a colorless liquid which sometimes acquires a greenish tinge from Fe and perhaps Ni; exposed to air it becomes brownish yellow, which is attributed by Betts[1] to the glue present. In working, the composition of the electrolyte is changed by becoming richer in lead and poorer in acid. The normal acidity is maintained by additions of fresh acid. Should the electrolyte become impoverished in lead, it is run over the granulated lead, as is ordinarily the H_2SiF_6. The temperature of the bath is maintained at 35 to 36° C. Temperature variations within considerable limits have no effect upon the character of the deposit, but, of course, affect the conductivity of the electrolyte. They also influence the asphalt coating of the tank, which is likely to crack if the temperature is too low and to blister and flow if it is too high. The head-tank contains copper coils through which steam is passed in winter, and cooling water in summer.

The circulation of the electrolyte in the tanks, arranged in double cascades, is at the rate of 3 to 4 gal. per min. with impure, and rises to 7 gal., with pure anodes. The centrifugal pumps which raise the electrolyte from the sump to the head are of copper and have bronze shafts. The loss in electrolyte is from 5 to 10 lb. H_2SiF_6 per ton of normal lead bullion. It is caused in part by dissociation of the acid, which increases with the impurities present in the anode.

292. Current.—The current density varies with the purity of the anode. Betts[2] worked with 4 amp. per sq. ft. in electrolyzing an anode containing Pb 66, Bi 7, Sb 19, As 5, Ag 2, Cu 1 per cent. Refineries aim to have anodes with about 98 per cent. Pb; with such material the current density varies from 16 to 18 amp. per sq. ft. anode area. The fall in potential from anode to cathode with an electrode distance of about 1¾ in. is from 0.35 to 0.55 volt, and varies with the current density.

The electrochemical equivalent of lead[3] is 103.43; the amount of lead transported per amp.-hr. is 3.857 gr.[4] The current efficiency in a plant is about 90 per cent. Calculations regarding the process have been carried out by Richards.[5]

293. Anode.—The lead bullion usually does not contain less than 98 per cent. Pb; of the remaining 2 per cent., Sb accounts for 1 to 1.25 per cent. Lead bullion containing more than 2 to 2.25 per cent. foreign metals causes complications in electrolysis. The anodes have shoulders which rest on the busbars of the tanks. A stationary cast-iron mold for anodes with shoulders is shown in Figs. 681–683. The anode has beveled sides and bottom; it is 20 and 21½ in. wide, 23⅜ and 24⅛ in. long, its thickness is 1½ in., its weight 350 lb. The mold has at the upper end a closely-fitting removable block, to permit the insertion of a two-prong pry-bar, Fig. 683*a*, for loosening the anode. An anode, 24

[1] Treatise, page 43.

[2] Treatise, page 56.

[3] Betts-Kern, *Tr. Am. Electrochem. Soc.*, 1904, IV, 67.

[4] Clarke, *Journ. Am. Chem. Soc.*, 1906, XXVIII, 307.

[5] Richards, J. W., "Metallurgical Calculations," McGraw-Hill Book Co., New York, 1918, 602.

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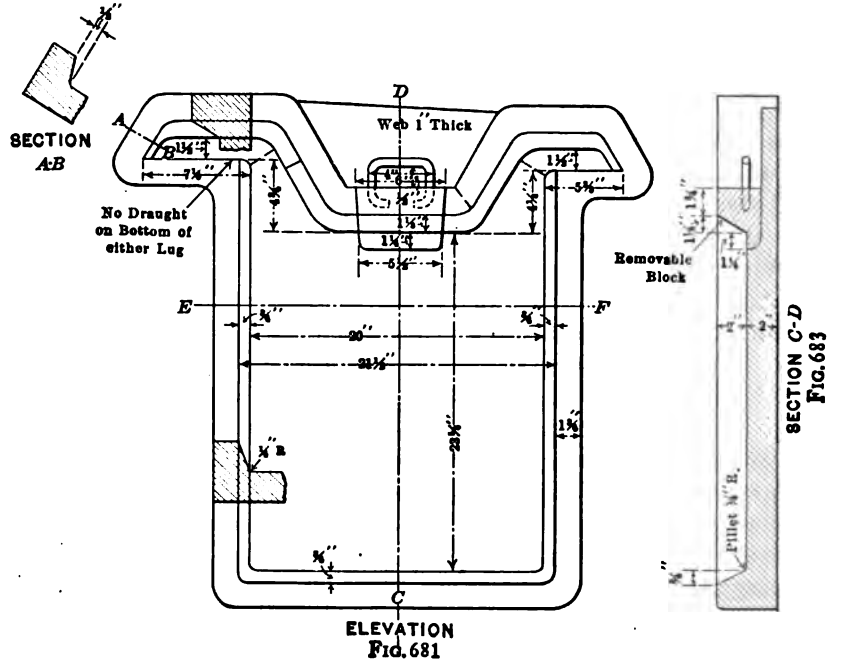
by 36 in. and 1½ in. thick, weighs 375 lb. The anodes are somewhat thinner in comparison with those used in copper refining, as during electrolysis the large amount of impurity present adheres to the anode and increases the resistance to the current. If the coating becomes too thick, it falls to the bottom and is likely to cause short circuiting; further, constituents of the mud are likely to go into solution or to be held mechanically in suspension, and to be deposited on the cathode.



FIGS. 681 to 683.—Stationary anode mold.

Anodes are cast in open molds. The upright Truswell closed mold,[1] used at first at Trail, was unsatisfactory, as the tops frequently were porous. The lead is cast from the kettle in which the blast furnace product has been freed from dross. It is done by means of a siphon or a pump delivering into molds placed either on a stationary frame in a three-quarters circle or on a rotating table. A stationary mold has been shown in Figs. 681–683. The Miller casting plant with rotating table, in use at Trail, B. C., is shown in Figs. 684–685

[1] *Eng. Min. J.*, 1906, LXXXI, 853.

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The leading parts are an oil-fired kettle with Miller centrifugal pump, a rotating table, driven by a motor through gearing and worm-wheel, which carries tilting anode molds, and a latch-tripper to invert the molds, and dump the anodes. The lead bullion from the blast furnaces is poured into two kettles, where it cools; it is drossed and pumped into the storage kettle of the machine. The kettle is 8 ft. 9½ in. in diameter and 4 ft. deep. In the center is the submerged centrifugal pump which delivers the lead through the inclined pipe into a horizontal branch provided with a riser and pipe-handle at the ends, and a tilting delivery-pipe in the center. The lead raised by the pump flows under a given head through the delivery pipe into the mold, if in the position shown in the figure. As soon as the mold is filled, the delivery-pipe is turned up, and the lead flows back into the kettle. In order to secure anodes of uniform thickness, a small 4-leg iron table with incandescent lamp (not shown) is placed on the empty mold. When the lead has risen in the mold to the required depth, it comes in contact with the poles of the lamp, closes the electric circuit, and causes the current to pass through the lamp which suddenly brightens. In passing from the feed to discharge-end, the lead in the molds is sprayed and solidifies. The anode is dumped. The mold is split under the shoulders and the lower part pivoted. Thus, in tilting the lower part of the mold, the shoulders of the anode are released from the mold; the anode is not bent by this procedure. The dumped anodes are taken from the dumper stand by means of a traveling compressed-air lift and deposited on to a transfer-car. The machine makes 0.151–0.187 r.p.m., casts per hr. 255–420 bars weighing 98–112 lb., and requires 3–4 h.p.

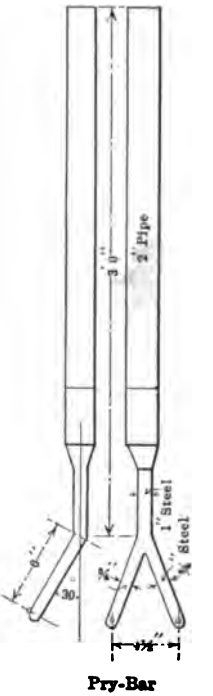

FIG. 683*a*.—Two-prong pry-bar.

294. Cathode.—This is about ½ in. larger all round than the anode. At first, the lead cathodes were plated sheet iron; later electro-deposited sheets of lead were used; at present they are cast with the apparatus devised by J. F. Miller, Trail, B. C., and shown in Figs. 686–688. A sloping cast-iron table, carried by an adjustable steel frame, has pivoted at the upper end a trough. The trough is filled from a kettle with cathode lead by means of a ladle, and tilted on to the table, whereupon the lead flows down the table; most of it solidifies in a thin, even plate, and the rest runs off the bottom or is caught in the gutters placed on the sides. The sheets are taken off from the table, piled, and later wrapped by hand around the cross-bars. Three men will cast 1100–1400 sheets in 6 hr.

An improvement on the original machine is represented in Figs. 689–690. It consists in casting fastening lugs at the same time that the plate is being cast, and is accomplished by providing the lower part of the inclined table with two tongues. The lead flowing down will form three strips which when

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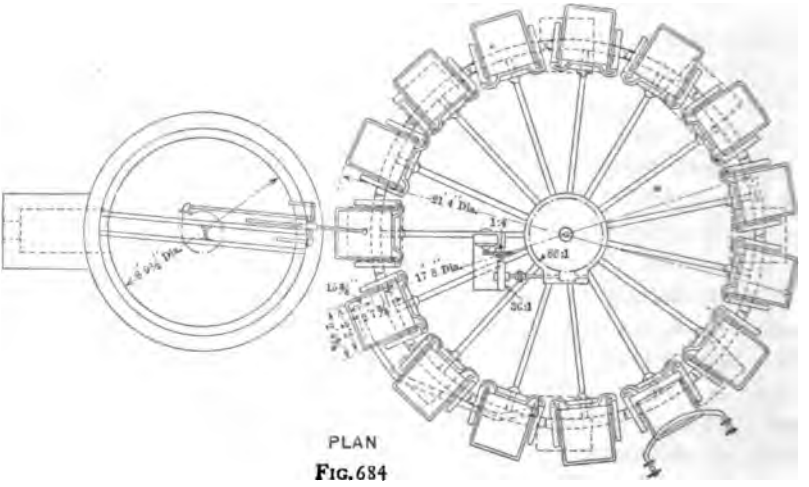
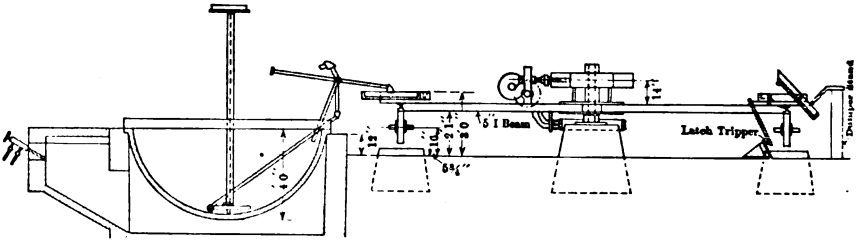
bent over a stick and welded each at two points by an arc burner, will furnish three loops through which is slipped the copper cross-bar. The sheets when removed from the table are flattened with wooden mallets.

Instead of having two projecting tongues on the plate, the spaces occupied by them have been cut out, *e.g.*, the table, 4 ft. 4 in. by 2 ft. 2 in., furnishing sheets 4 ft. 1 in. long, has two slots, $2\frac{1}{8}$ by $6\frac{1}{4}$ in. which begin 5 in. from the lower edge of the table. The lead flowing down the table forms three strips $11\frac{3}{4}$ in. long, as it does not cover the part of the table just below the slots.



PLAN

FIG. 684



SECTIONAL ELEVATION

FIG. 685

FIGS. 684 and 685.—Miller anode-casting machine.

295. Electrode Distance.—This is $1\frac{3}{4}$–$2\frac{1}{2}$ in.; in the plants named it ranges from $2\frac{1}{10}$ to $2\frac{1}{4}$ in., center to center, and is about $1\frac{5}{8}$ in. face to face. In all refining work, a distance that is small decreases resistance and increases kw. output, but tends to cause bridging of space and thereby short-circuiting with consequent loss in ampere efficiency; one that is large, increases the resistance, decreases kw. output, but generally gives a better ampere efficiency. The proper distance has to be determined for prevailing conditions.

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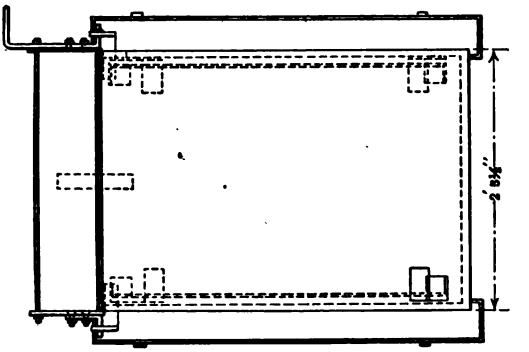
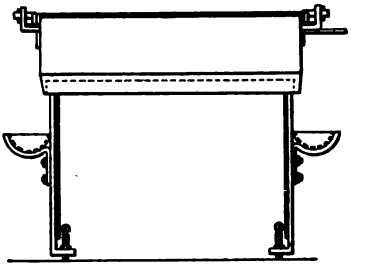
296. Depositing Vat.—The vats used for a number of years had the same form as those common with copper, that is, rectangular tanks with flat bottoms; they were and are coated with asphalt to a thickness of ¼ in., a heavier layer having a tendency to crack. At present petroleum residue asphalt is replacing the older natural asphalt, as it has been found not to crack when too cool, nor to blister when too warm.

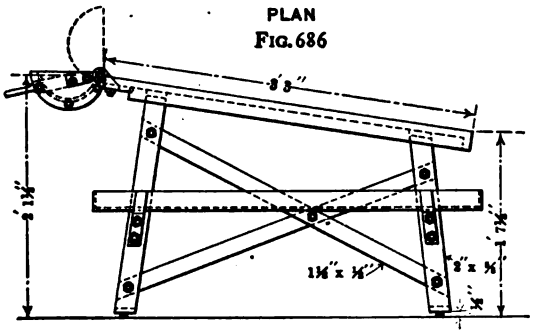
The early tanks at Trail, B. C., arranged in cascades, were 7½ ft. long, 2 ft. 6 in. wide and 3 ft. 6 in. deep, inside measure; they were made of 2-in. fir



PLAN
FIG. 686



END ELEVATION
FIG. 688



SIDE ELEVATION
FIG. 687

FIGS. 686 to 688.—Miller cathode-casting machine.

planks, bolted together, nailed with copper nails, and braced, ends and sides, with 3- by ½-in. iron bands; the wood was protected by P and B paint; the circulation was obtained through 1½-in. hard rubber pipes. The first 28 tanks were arranged in 2 rows of 14 each, the tanks being placed in pairs as shown in Fig. 691, the current entering and leaving through copper busbars 4 to 1¼ in. The additional 44 tanks were arranged side by side, as shown in Fig. 692, 22 in a row, each tank being connected with the next one ahead by a sheet of copper 8×1/32 in.; the current entered and left through busbars 4×¼ in.; the electrolyte was made to pass through 2 tanks instead of through 14 as in the first tanks. The advantage of this mode of circulation over the first is shown

[1] Whitehead, *Mines & Minerals*, 1905, XXV, 285.

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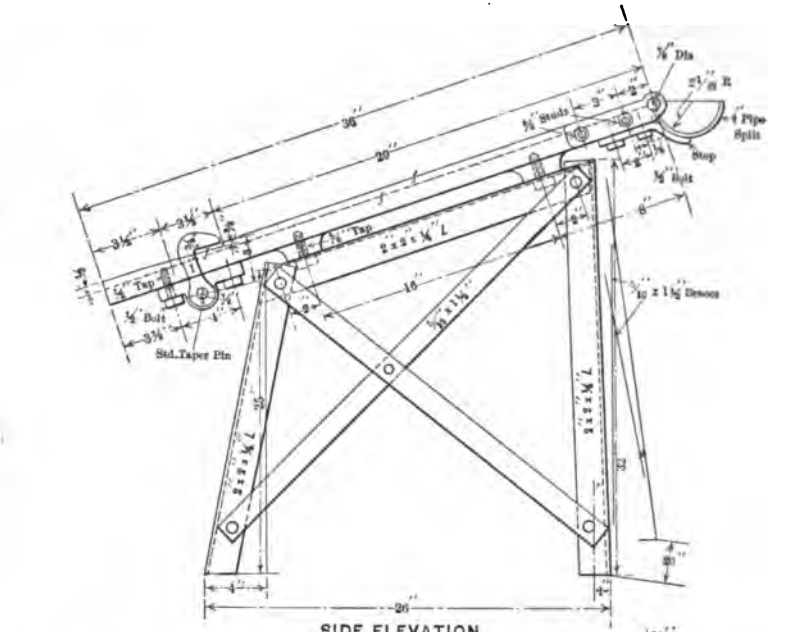

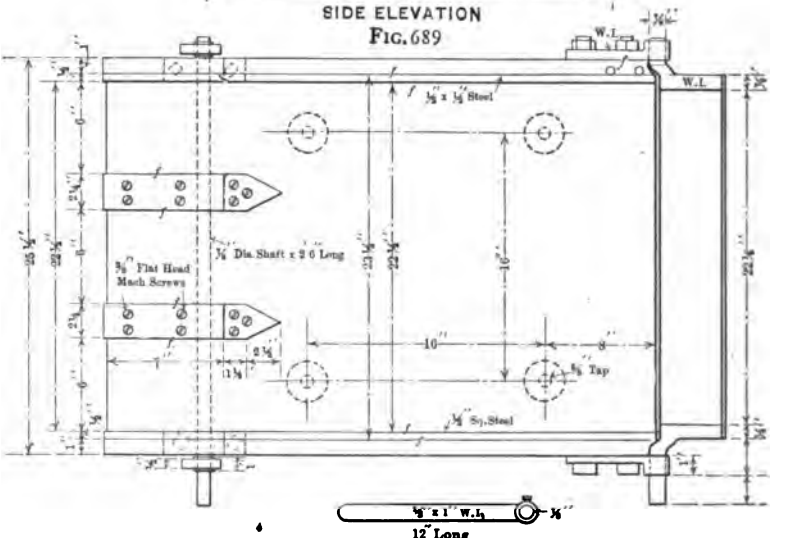

FIGS. 689 and 690.—Improved cathode-casting machine.

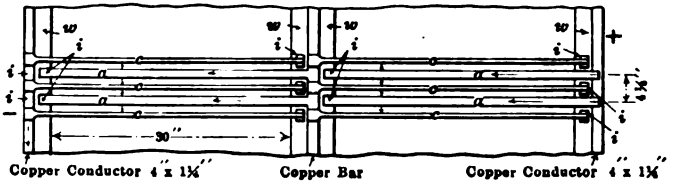

FIG. 691.—Electrolyzing vats in two rows, Trail, B. C., 1905.

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in Table 151 which brings out the tendency of the electrolyte to form a heavy layer on the bottom of a tank if the travel is not sufficiently swift or its path is not properly governed.

TABLE 151.—CIRCULATION OF ELECTROLYTE

FLOW THROUGH 14 TANKS

	Beaumé	Specific gravity	Lead per cent.	Total acid per cent.
Top	20½	1.167	4.03	12.52
Middle	21	1.171	4.23	12.60
Bottom	23	1.19	5.24	13.13

FLOW THROUGH 2 TANKS

	Beaumé	Specific gravity	Lead per cent.	Total acid per cent.
Top	23	1.19	4.8	13.69
Bottom	24	1.2	4.95	13.71

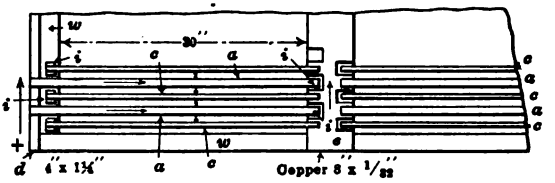

FIG. 692.—Electrolyzing vats, side by side, Trail, B. C., 1906.

The new tanks at Trail are built of cement concrete which receives a coat of P and B paint; the latter is protected from any action of the electrolyte by a layer of asphalt ¼ in. thick. The leading features of the plant are shown in Fig. 693, and those of the vats in Figs. 694–696. The plant has 408 cells, produces daily 100 tons refined lead, and receives a current of 3000–3500 amp. at 90 volts. The cement vats are arranged in cascades with a drop of 3 in. between tanks. The electrolyte is circulated through 1-in. hard rubber pipes with a flow of from 4 to 6 gal. per min. The pipes delivering and withdrawing the solution of a tank are placed diagonally at the ends so as to counteract layering; the withdrawing-pipes extend to 6 in. from the bottom. Centrifugal pumps of copper raise the electrolyte to the head tank.

A tank, 7 ft. 4½ in. by 2 ft. 6½ in. and 3 ft. 8 in. deep, holds 20 anodes and 21 cathodes, which leave a bottom-space 20 in. deep; the electrode distance is 2 1/10 in. The anodes, 3 by 2 ft. and ½ in. thick, weigh 375 lb., and assay Pb 98 per cent. and Sb 0.3–1 per cent.; the cathodes, 3 ft. ½ in. by 2 ft. 1 in. and ¾ in. thick, weighing 150 lb., are wrapped around copper cross-bars, ½ × 1 in., flattened at the ends; the busbars, 3 × 2 in., are not in contact with the concrete walls, but rest on ¼-in. wooden insulators coated with paraffine paint.

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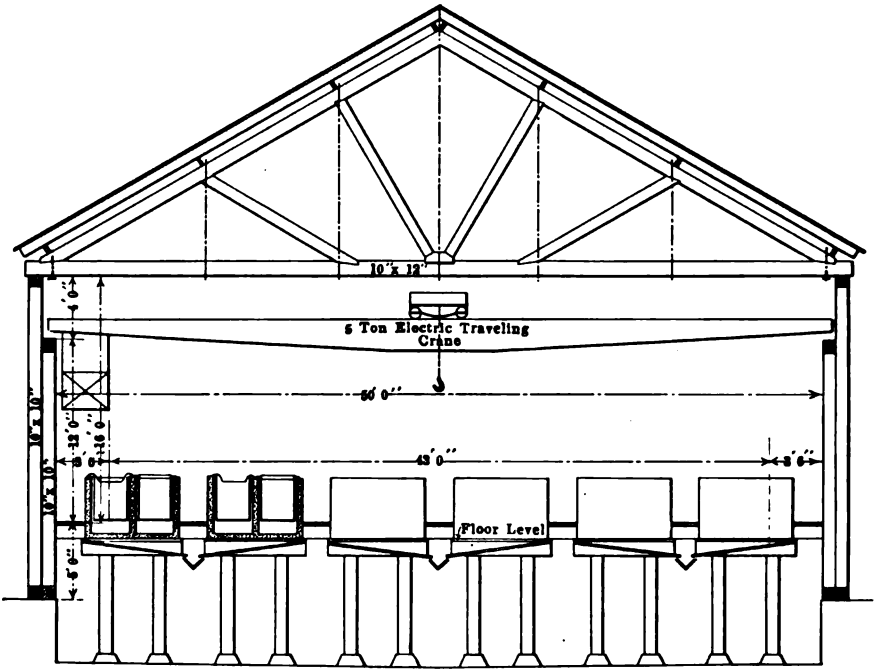
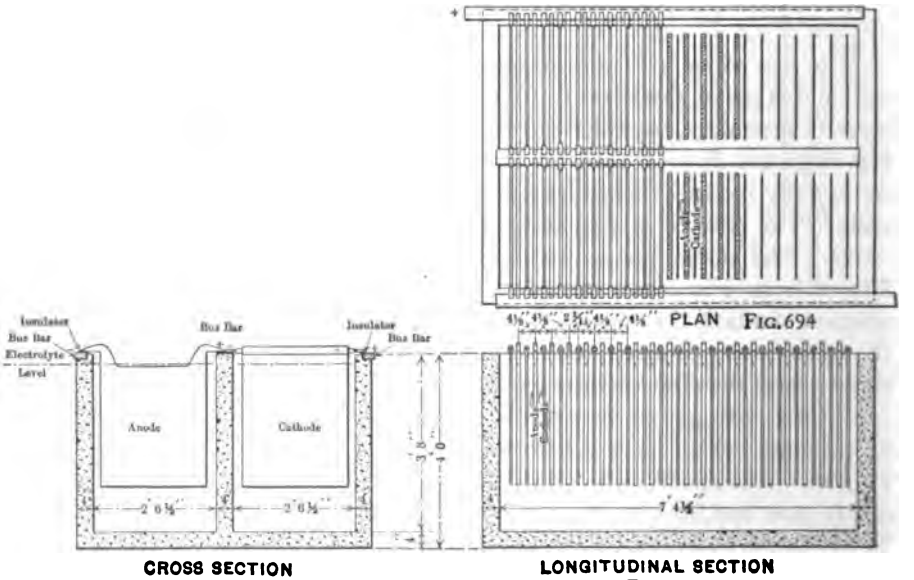


FIG. 693.—Tank-house, Trail, B. C.



FIGS. 694 to 696.—Electrolyzing vats, Trail, B. C.

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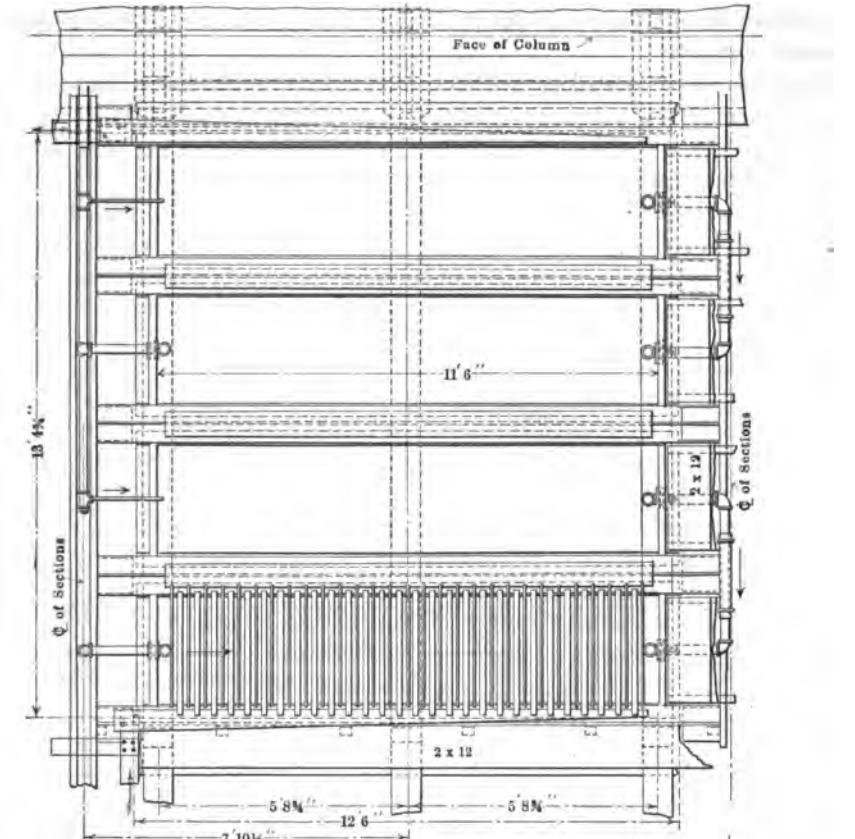

FIG. 697 PLAN

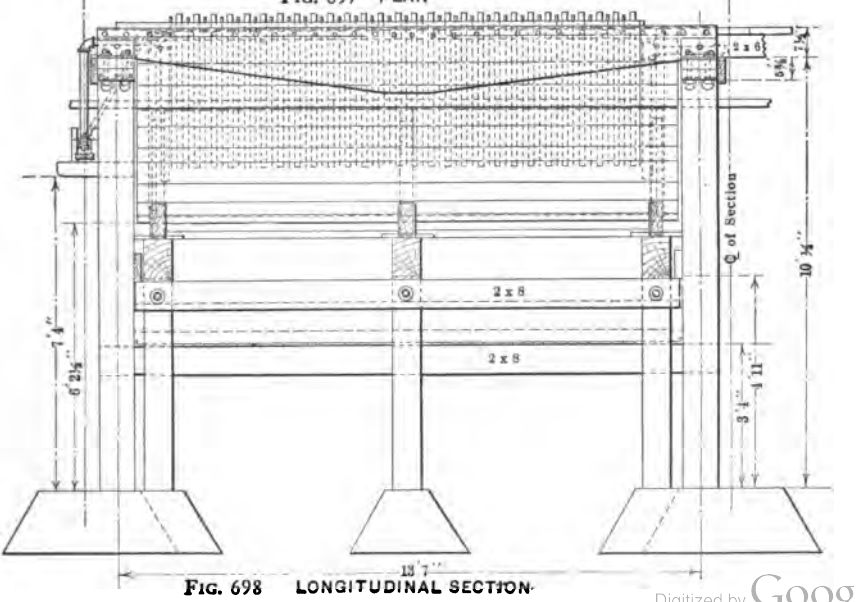

FIG. 698 LONGITUDINAL SECTION

FIGS. 697 and 698.—Electrolyzing vats, East Chicago, Ind.

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The current density is 14–16 amp. per sq. ft. cathode area, the drop in potential per vat is 0.3 volt.

Electrodes are handled by an electric crane, a tank-load at a time; anodes are exchanged every 5 days and give 20–22 per cent. scrap, cathodes every 4–5 days. The electrolyte contains 4–5 per cent. Pb and 10–11 per cent. H_2SiF_6, and has a temperature of 25–30° C.

The refinery of the U. S. Metals Refining Co., at East Chicago, Ind., has 208 vats, arranged in 52 units of 4 cells; it works with a current of from 5400 to 5800 amp. at 100 to 115 volts, and produces in 24 hours 100 tons refined lead. The Walker system of arrangement of tanks in units of four, shown in

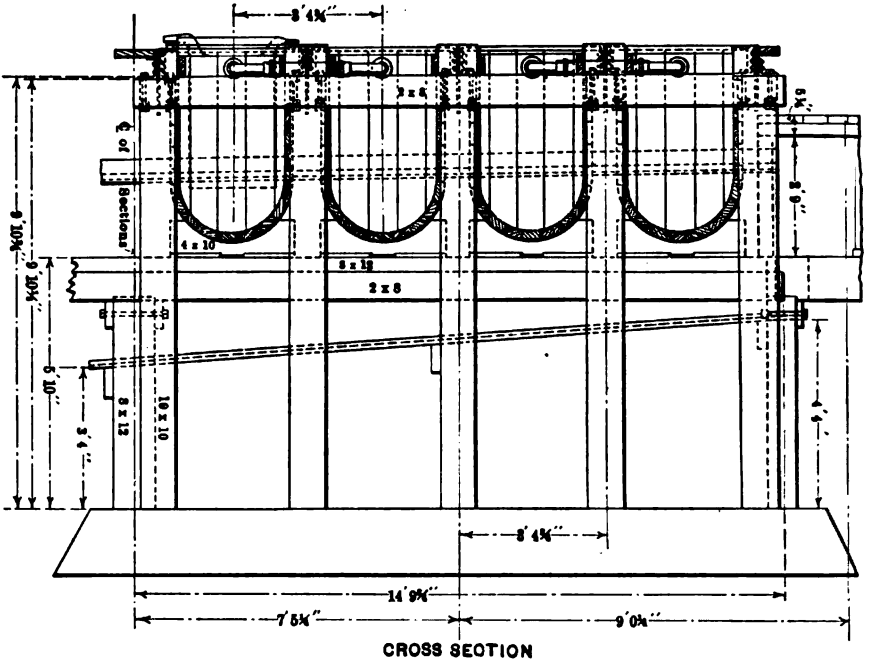

FIG. 699.—Electrolyzihg vats, East Chicago, Ind.

Figs. 697–699, is in operation for the 44 units originally installed. The new round-bottom tanks, which are replacing the older flat-bottom forms are also of wood, but of the Thum pattern.[1] Their characteristics are, that they do not carry the electrodes, which are suspended independently, and, being relieved from pressure, can have rounded bottoms of stave construction held together with rod-hoops. The tanks, 2 ft. 6 in. by 11 ft. 6 in. and 4 ft. deep, are of 2–2¼-in. pine (the older flat bottom tanks have 5-in. sides) and therefore light; they are kept tight by rod-hoops. A tank stands insulated by glass plates on 3 wooden supports, 4 by 10 in. and 2 ft. 10 in. long, which are carried by cross beams. Posts, tied by braces, 2 by 8 in. and 15 ft. long, carry steel

[1] U. S. Patent No. 10957448, May 15, 1914.

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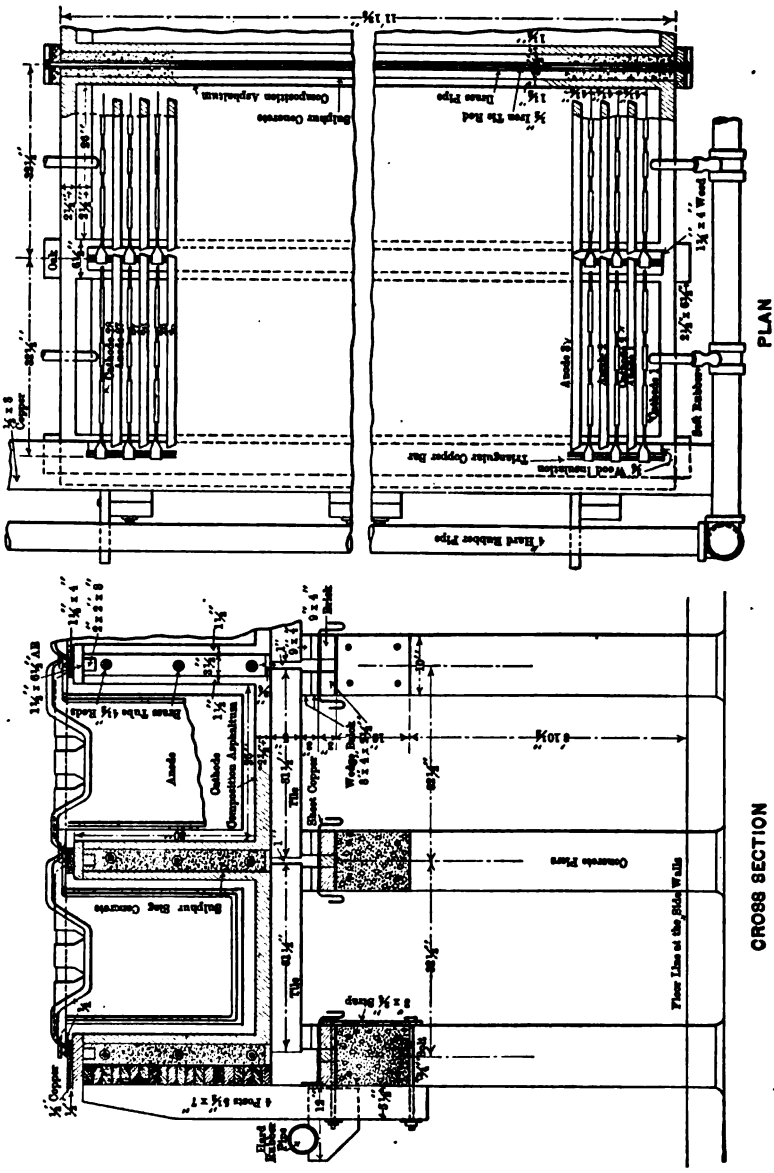
trusses on which rest insulated copper busbars, ⅛ by 10½ in. and 11 ft. 2 in. long. A tank, coated with pitch from oil-residue in the usual way, holds 28 anodes and 29 cathodes. The anodes, 24 by 36 in. and 1⅛–¼ in. thick, weigh-



FIGS. 700 and 701.—Electrolyzing vats, Omaha, Neb.

ing approximately 460 lb., are suspended from shoulders; the cathodes, 1 in. larger than the anodes, are suspended each by 3 strips of lead from cathode bars 2 ft. 9¾ in. long and ½ in. thick. The electrolyte, with 10 per cent. Pb and 12 per cent. H_2SiF_6 is kept at 35–38° C., circulated in the direction of the

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arrows through a block of 4 tanks at a rate of 3–4 gal. per min. by means of hard rubber pipes. Submerged copper centrifugal pumps with bronze shafts raise it to the head tank; the troughs are of wood coated with pitch. The current density is 17–18 amp. per sq. ft.; the fall in potential per vat is 0.35–0.50 volt.

The Betts department of the refinery of the American Smelting and Refining Co. at Omaha, Neb., confines its work to the treatment of anodes with about 1.5 per cent. Bi. It has 80 tanks in series, and 20 cascades, or 4 tanks connected up according to the Walker system,[1] treated in 1917 18,000 tons lead. Details of tanks and electrodes are shown in Figs. 700–702. The tanks are of asphalt concrete, made up of a mixture asphalt, asbestos and sand; the spaces between adjoining tanks are filled with concrete or crushed slag and poured sulphur. The tanks rest on square glazed drain-pipes; these are supported by two courses of brick, the upper of wedge and the lower of straight brick; between the courses are placed sheets of copper with troughs to carry off drainage. The lower bricks are separated by asphalt blocks. The supporting bricks are carried by reinforced concrete beams, and these by concrete piers. The ends of a block are braced by wooden buck stays tied with ½-in. rods passing in brass pipes through the slag-sulphur concrete spaces; the sides are reinforced by 4-in. walls built of wooden strips 2 in. thick, and wooden posts anchored in the concrete beams. A tank, 10 ft. ⅜ in. long, 26 in. wide and 30 in. deep, holds 27 anodes and 28 cathodes. The anode has beveled sides; it is 20½ and 21½ in. wide, 23⅜ and 24⅛ in. long, 1½ in. thick, and weighs 350 lb.; the cathode is 1 in. wider and longer than the anode, is ⅛ in. thick, and weighs 6 lb.; the finished cathode about 130 lb. The tank walls are covered with insulated oak planks, 1½×6 in., which carry primary busbars, ½×8 in., and secondary, 1¼×4 in. Anodes are exchanged through an overhead electric crane in tank-lots every 11 days, cathodes every 5 or 6 days; the anode scrap amounts to 20–25 per cent.

297. Corrosion of Anode and Deposition on Cathode.—The corrosion of the anode is, on the whole, regular, as the amount of impurity is comparatively small and practically all of it insoluble. It has been found necessary to allow from 20 to 28 per cent. of the anode to remain uncorroded, which goes as scrap to the anode kettles. The reason for this is that it is necessary to provide a full unbroken anode surface for the mud to cling to. With the usual grade of lead bullion only a small amount of mud is detached from the anode. If the mud exceeds a certain thickness it glides down from the anode or particles become detached and collect on the bottom. This is likely to cause short circuiting as well as secondary chemical reactions between the components of the mud and the free acid of the electrolyte. Most plants clean up every 10 or 11 days, which coincides with the renewal of the anodes. A tank-load of corroded anodes, with the mud adhering to most of them, is raised from the vat, and deposited in a tank-car, twice the size of the cell, filled with water. The anode mud is scraped off, and the anodes are brushed by hand or with

[1] Hofman, "Copper," 1918, 532.

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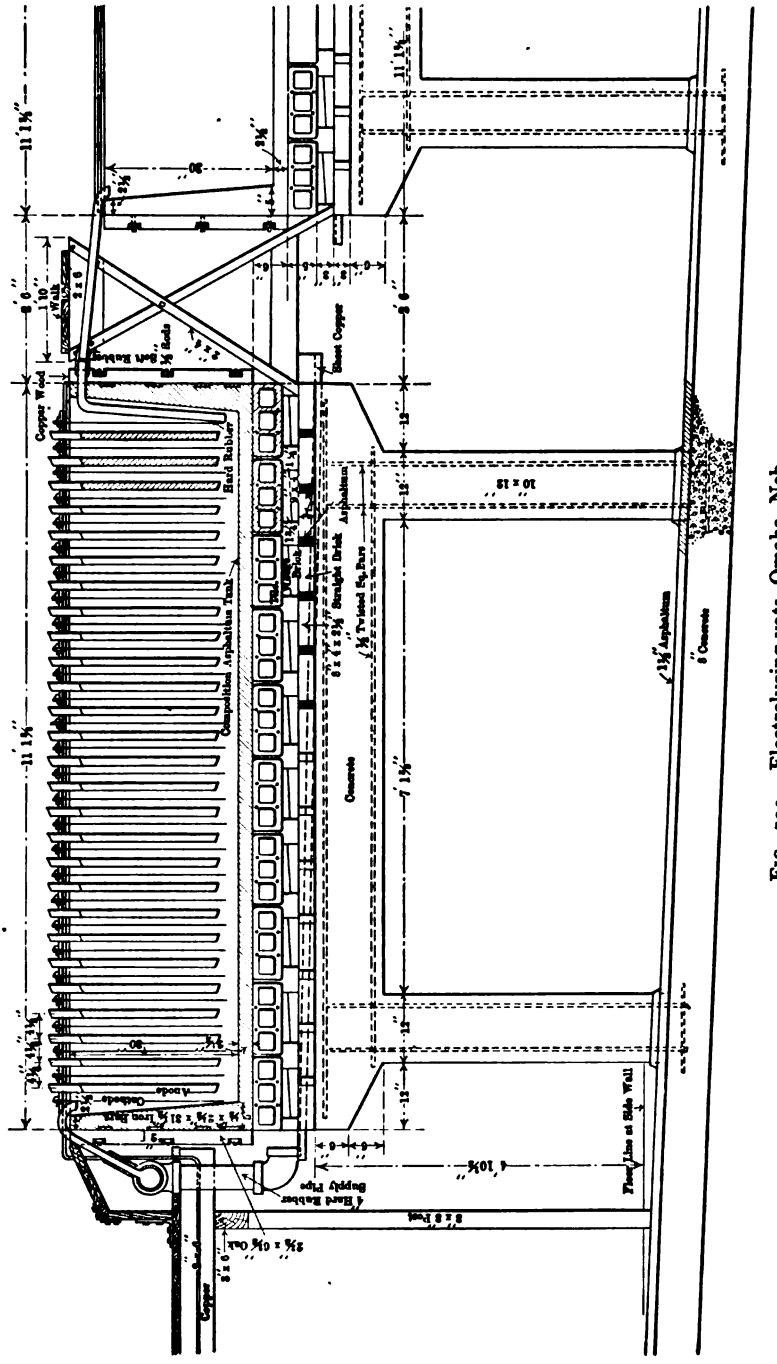

FIG. 702.—Electrolyzing vats, Omaha, Neb.

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Thum rotary brushes, using an electric hoist to raise and lower the anodes; when clean, they are raised, rinsed, and transferred to the anode kettles. The mud is partly settled, pumped with remaining electrolyte through a filter press, washed with water, dried with air or other means, and discharged when ready for further treatment. A bronze (Cu 95, Sn 5) Johnson or Schriver press, 18×18 in. with 12 leaves, furnishes 1-in. cakes, which retain 30 per cent. water and weigh 400–500 lb. The electrolyte in the pores of the slime takes up some lead which is not removed by the wash-water. The first (strong) filtrate, consisting of electrolyte, goes to the head tank; the dilute weak portion is used for first water in the anode wash-tanks. Most of the soluble lead is removed by filtering, but the mud retains about 10 per cent. insoluble lead.

The cathode deposit under good working conditions is smooth, but otherwise rough and streaked; at the edges it is thicker than at the center; and frequently knobs are formed. The cathodes are also removed by tankloads to a washing tank, sprayed with water, and transferred on a car to the melting kettles. Dilute solutions are evaporated.

298. Anode Mud.—The anode mud obtained from the corroded anodes and collected from the tanks varies in composition as seen in Table 152.

TABLE 152.—COMPOSITION OF ANODE MUD

	Ag	Au	Pb	Sb	As	Bi	Sn	Cu	H_2O	Fe	Se	Te
Trail, B. C.....	4	0.2	3–7	32	0.5–3.0	None	tr.	7	.....	None	None	....
E. Chicago, Ind....	15	0.7	10–15	35–40	5–8	5–8	1–1.5	2–4	.....	0.1–0.2	0.20	1–2
Omaha, Neb.....	10	...	2.7	10	0.2	30	.....	3	40	n.d.	n.d.	n.d.

More complete analyses have been published by Betts,[1] Whitehead,[2] and Kern.[3]

The treatment of the mud is governed by the presence or absence of Bi and with it of Te and Se; it is comparatively simple if Bi, Te, and Se are absent; it is complicated if Bi is to be recovered.

Whatever may be the treatment, the mud coming from a filter-press or suction filter is somewhat air-dried, transferred to iron trays on iron cars, wheeled into the flue of the reverberatory smelting furnace, or into a connected drying chamber, and allowed to remain there for from 24 to 48 hours that the moisture may be driven off and the metals more or less oxidized.

The anode mud from Trail, B. C., is free from Bi and Te. The oxidized mud is melted in the coal-fired reverberatory furnace shown in Figs. 703–705, the hearth of which is lined with magnesite brick. From 20 to 25 tons of partly oxidized mud are smelted in about 10 days without the use of any flux, the antimony slag being raked off as soon as the hearth is filled. When toward the

[1] *Tr. A. I. M. E.*, 1904, XXXIV, 182, 183.
[2] *Mines and Minerals*, 1905, XXV, 288.
[3] *Met. Chem. Eng.*, 1911, IX, 417.

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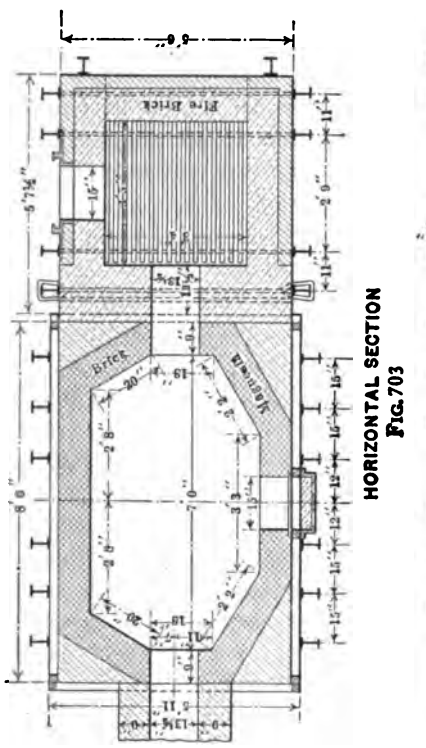
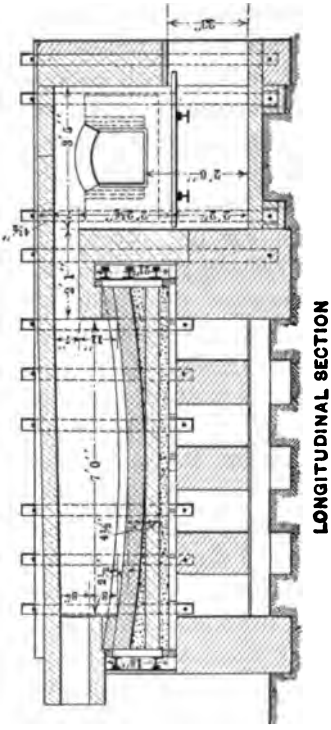
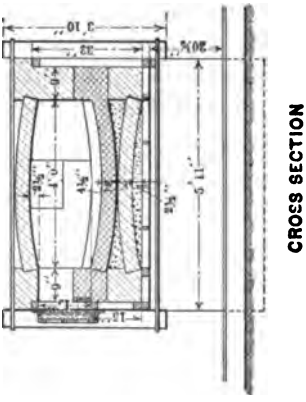
end of a campaign, the last of the antimony slag has been removed, there rises to the surface of the doré silver a crust consisting mainly of Cu_2O; this is skimmed, and the doré silver, from 80,000 to 100,000 oz., dipped. The antimony



HORIZONTAL SECTION
FIG. 703



LONGITUDINAL SECTION
FIG. 704



CROSS SECTION
FIG. 705

FIGS. 703 to 705.—Reverberatory furnace for anode mud. Trail, B. C.

slag, which assays from 300 to 500 oz. doré per ton, is resmelted once or twice with fine coal in the reverberatory furnace, whereby its content in precious metal is reduced to about 8 oz. Ag per ton. The purified antimony slag then goes to the blast furnace to be smelted for hard lead.

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In 1908 A. J. McNab[1] worked out at Trail, B. C., a wet process for the treatment of the anode mud, which consisted of the following steps:

(1) Solution of Sb, As, Te, and Se in sodium polysulphide and electro-deposition of antimony from the filtered solution; accumulating arsenic is to be removed by concentration of liquor to 35° Bé. and subsequent cooling. (2) Sulphatizing roast of the residue (Ag, Pb, Bi, Cu, Au); extraction of copper (also some Ag and Bi) with water and sulphuric acid; precipitation of silver and bismuth with copper. (3) Fusion of the residue (Au, Ag, Pb, Bi) from leaching the sulphatized material, in a reverberatory furnace; separation of lead and bismuth from gold and silver by cupellation; recovery of bismuth by the known methods; concentration of the copper solution, etc.

The process was put into operation, but given up again, when the Sb in the hard lead brought the same price as if unalloyed with lead.

The treatment of anode mud containing Bi has been worked out at the works of the U. S. Metals Refining Co., East Chicago, Ind., and is used in a modified form at Omaha, Neb.

Details are reserved for the present; they will be discussed in a later work in connection with the metallurgy of bismuth.

299. Comparison Betts and Parkes Processes.—It has been stated that there exist only four plants which use the Betts, and that in one of these it forms a subordinate department for the treatment of lead bullion rich in Bi. The advantages of the Betts process are that it furnishes a high yield of refined lead that is free from Bi, and a means of recovering this metal. The disadvantages are that the refined lead contains 0.20 to 0.30 oz. Ag per ton *versus* 0.10 to 0.12 oz. of the Parkes lead, and that the cost of electrolysis is 50 per cent. more than that of zinc desilverization. In order that the Betts process may compete with the Parkes, it is essential that the Bi-content of the lead bullion at least make good the difference in cost of treatment. The Bi-content of the usual run of lead bullion is very low. It is the common practice of Parkes plants to sacrifice the Bi and to make up softening-furnace charges which contain less than 0.05 per cent. Bi, and thus produce a refined lead which meets the requirements of corroding lead.

[1] U. S. Patent No. 907754, Dec. 1, 1908.

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CHAPTER XI

LEAD POISONING[1]

300. General.—It is the universal experience that men engaged in working lead ores, metallic lead or lead compounds are more or less affected by the poisonous character of the materials they are handling; they are "leaded," *i.e.*, suffer from plumbism. Though men and women who make preparations of lead are by far the greatest sufferers, the lead smelteries and refineries have poisoned sufficient employees to give them a bad name. The earlier plants in the United States have been more careless in this regard than have the older of Europe, where government supervision has been more exacting; but in the United States the protection of workmen against poisoning has made considerable progress, so that the number of men afflicted at present is small in comparison with that of even 10 years ago.

The lead plant has as its raw material the ore, which is sampled, roasted, and smelted. Sampling makes dust; roasting and smelting make both dust and fume. The end-product of the smelting process, the lead or lead bullion, and the intermediary products, speiss, matte, slag, flue-dust, and fume, are subjected to fire-processes, in all of which there are again formed poisonous dust and fume. It is not possible to carry on these operations without making dust and fume, hence both finely divided products have to be removed as soon as formed and thus brought out of contact with the men who are producing them. When the collected dust and fume have to be handled, the men doing this work have to be protected by special devices, so that the fine materials cannot enter their systems and exert harmful effects. It was formerly held that lead was absorbed by the skin; this, it is now believed, does not occur as long as the skin is not broken. Lead enters the human body through the mouth and the

[1] Fuller, M. Meissner, C., "Hygiene der Berg-, Tunn- und Hüttenarbeiter," Fischer, Jena, 1895.

Report of Austrian Secretary of Commerce, "Bleivergiftungen in Hüttenmännischen und Gewerblichen Betrieben, Ursachen und Bekämpfung," 8 parts, Hölder, Vienna, 1905-1913.

Müller, R., "Die Bekämpfung der Bleigefahr in Bleihütten," Fischer, Jena, 1901.

Leyman, "Die Bekämpfung der Bleigefahr in der Industrie," Fiseher, Jena, 1908.

Rambousek, J.-Legge, T. M., "Industrial Poisoning," Arnold, London, 1913.

Collins, E. L., "H. M. Medical Inspector of Factories," Special Report on Dangerous or Injurious Processes in the Smelting of Materials Containing Lead," London, 1910.

Clifton, Industrial Lead Poisoning, *Min. Sc. Press*, 1912, CV, 9.

Libert, J.-Firket, V., Conditions de salubrité intérieure des usines Belges, plomb et argent, *Ann. Min. Belg.*, 1913, XVIII, 449; reprint, Béranger, Paris-Liège.

Hamilton, A., "Lead Poisoning in Smelting and Refining of Lead," *Bull.* 141, U. S. Bur. Labor and Statistics, Washington, 1914.

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nose, is acted upon by gastric juices, and the dissolved lead forms the poison. The solvents in gastric juice are hydrochloric acid and peptone.

301. Lead and its Compounds as Poisons.—The boiling-point of metallic lead is 1525° C.; but the metal is volatile at much lower temperatures. The experiments of Lewin[1] have shown that at atmospheric pressure molten lead does not give off any vapor at from 500° to 520° C., that at 850 to 900° C. some is noticeable, and that the presence of other metals, especially zinc and antimony, causes vapor to form at 750° to 800° C. Lead vapor is quickly changed to oxide.

Lead sulphide melts at 1120° C., but is volatile at 600° to 800° C.; it is oxidized at 360° to 380° C. into oxide and sulphate.

Lead oxide is volatile at 883° C.

Lead sulphate is split at 900° C. into basic sulphate and sulphur trioxide.

Lead carbonate is dissociated at 315° C. into lead oxide and carbon dioxide.

Metallic lead is sparingly soluble in hydrochloric acid. Of the different compounds, lead sulphide is least soluble at body temperature in gastric juice; then follows lead sulphate, normal and basic; the oxide, both normal and basic carbonates, are the most soluble.

The lead in the dust and fume of smelteries is not an acute poison. Its effect is cumulative; small quantities taken into the body by daily contact with the source are absorbed by the system. They show their effects with some men in a week; with others it may take a month or months before they show any symptoms of being poisoned. The beginning of lead poisoning[2] is noticed by a disturbance of general health, a sense of weakness accompanied by a decline in strength, severe headaches, lack of appetite accompanied by great thirst, pain in the stomach and vomiting, bad breath, loss of color followed by sallow complexion. Discoloration of gums with blue line at contact of gums and teeth is usually noticed, but not always; ulceration of mucous membrane is common. Painful constipation (lead colic) and retention of urine are characteristic in acute attacks. In chronic cases, pain in limbs, trembling of fingers, swelling of joints, lameness of hands and legs, afflictions of eye-sight, smell, and taste, and other serious complications are of only too frequent accurrence.

302. Remedies.—The presence of dust and fume in a lead plant is unavoidable. The principle to be followed in correcting the evil is to collect dust and fume at the place where they are produced in as concentrated form as possible, that is, not to allow them to become diluted with air before an attempt is made to withdraw them. For this reason natural draft is less efficient than suction by means of fans. Fans are being used more and more; in many instances they have not been an expense, but even a source of profit. This became evident when the gases laden with dust and fume were delivered into separate bags of a bag house, and the collected materials valued.

[1] *Metall. u. Erz.*, 1913, X, 441.

[2] Sommerfeld, Th.-Fischer, R., transl. by Rand, W. H., "List of Industrial Poisons and Other Substances Injurious to Health, etc.," Ohio State Board of Health, Columbus, O., 1914, 21.

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Ores are received in lump form and as a concentrate. Carbonates make more dust than sulphides. All ores have to be weighed and sampled. Handling, crushing, and sampling are operations which make much dust, unless the ore is wet or moist. It will not do to moisten dry ores, as a moisture-sample has to be taken in order to arrive at the dry weight on which the assay is based. The remedy therefore lies in having the spaces for unloading well ventilated, (§121), in having fans at the places where ores are comminuted, in having machine-sampling whenever possible, in forcing sampling men to wear respirators when necessary, and taking the usual precautions given below, and in keeping the floors sprinkled and clean. Charging floors require the same attention as sampling floors.

In the discussion of the various apparatus for roasting, smelting, and refining, attention has been called to the modern equipments for withdrawing by means of fans all the fumes that are formed.

Some of the analyses of fumes from various sources collected by Iles[1] are given in Table 153.

All apparatus and devices are of little use, if the management of the plant does not strictly enforce their use. For this purpose it is advisable to lay down definite regulations which are to be followed rigidly by all employees. It is also necessary that the company have in its employ a physician who is conversant with the treatment of cases of lead-poisoning.

Some rules for the guidance of employers and employees are submitted.

Boys ought not to be employed, and men only when they have been passed upon by the physician.

Men who, soon after employment, show signs of lead-poisoning ought to be dismissed; men who have suffered from lead ought to be given temporary work which does not expose them to lead. The length of shift ought to be governed by the danger of the work.

Food ought to be brought in tight pails. Eating in the place of working ought to be forbidden; for this special rooms ought to be provided which contain washing appliances with free soap and towels. Washing of hands and face and rinsing of mouth ought to be insisted upon before anything is placed in the mouth. The mouth ought to be rinsed before taking a drink or tobacco, or after having being exposed to dust or fume.

In passing through dust or fumes, a man ought to hold his breath; if exposed for a short time, he ought to tie a pocket handkerchief over his nose and mouth; if for a longer time he ought to use a sponge or other respirator, furnished free by the company.

A man ought to have had a substantial meal containing fats before starting work; he should put a little vaseline up his nose. He should avoid alcohol and acid foods, and should drink freely water and milk.

He should be careful not to become constipated.

Any indication of sickness ought to be reported at once to the physician.

Room ought to be supplied for the change of clothing, and closets provided

[1] *School Min. Quart.*, 1899, xx, 397.

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TABLE 153.—ANALYSES OF LEAD FUME

Description	SiO_2	Fe	Mn	S	Zn	Cu	Pb, dry	As	Sb	Al_2O_3	Ag, oz. per ton	Au, oz. per ton
Fume from the hood over breast of cupelling furnace							41.80				134.30	0.32
Dust from cupelling furnace dust chamber							16.60				71.50	Trace
Dust from cupelling and concentrating furnace chamber							27.60				106.60	0.40
Dust near cupelling furnace, in flue							57.00				67.00	0.32
Dust from softening furnace flue							33.80				30.00	0.16
Mixed fume from softening and refining furnaces							20.80	7.06	4.28		5.00	Trace
White fume from outside door of softening furnace							20.60	2.56	32.73		7.00	0.05
Retort fume, base of stack, large sample							0.60				29.70	0.32
Retort fume, average of all dust in flue					12.20		0.50				16.80	0.20
Retort fume, collected on top of retort flue							Trace				38.50	Trace
Retort fume, collected on top of iron columns							Trace				44.00	Trace
Retort fume, settled on plank near retorts					25.20		1.30				44.00	1.12
Dust from the base of a 100-ft. refining stack							38.70				71.20	0.08
Fume from antimony furnace							38.30	5.04	2.88		7.20	Trace
Fume from softening furnace							19.20	0.38	10.45		32.50	Trace
Fume from cupelling, concentrating, and silver melting furnaces					0.6		72.50	N.	Trace		85.90	0.10
Fume from softening, refining, and antimony furnaces							28.00				288.00	0.36

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for the working and street clothes. Working clothes ought to be washed at stated intervals. Bathing ought to be free, and ought to be encouraged, including the washing of the head.

As to treatment, the aim is to facilitate the elimination of lead through the kidneys and the intestinal canal. Potassium iodide, which was commonly used, has been found to have no beneficial effect. Hutchings found lozenges made up of sulphur, sugar, and peppermint, and containing 0.5 grain of sodium sulphate, to be efficacious at the works of Cookson & Co., Newcastle-upon-Tyne, which produce daily 200 tons white lead and 50 tons red lead.[1]

The use of drinking water slightly acidulated with sulphuric acid has been advocated; also drinking water charged with table salt; but definite results have not been recorded. A novel treatment which has been tried recently is by the Clague Electrolytic System,[2] in which a direct current is passed through a man whose bare feet and hands are placed in cells, charged with brine, and connected by electrodes with a current of 60 milliamperes at about 20 volts. At the Selby Lead Works, Selby, Cal.,[3] daily treatment for 30 min. on 30 consecutive days proved to be efficacious. No lead, however, was found in the electrolyte.

[1] Blum, F., "Untersuchungen über Bleivergiftung und ihre Verhütung in Industriellen Betrieben," Frankfort s/M., 1900.

[2] Oliver, *Royal Society of Edinburgh*, through *Lancet*, 1914, 1, 853.

[3] E. N. Engelhardt, letter, Sept. 12, 1917.

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